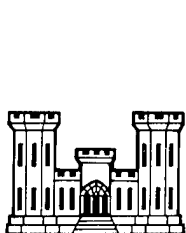


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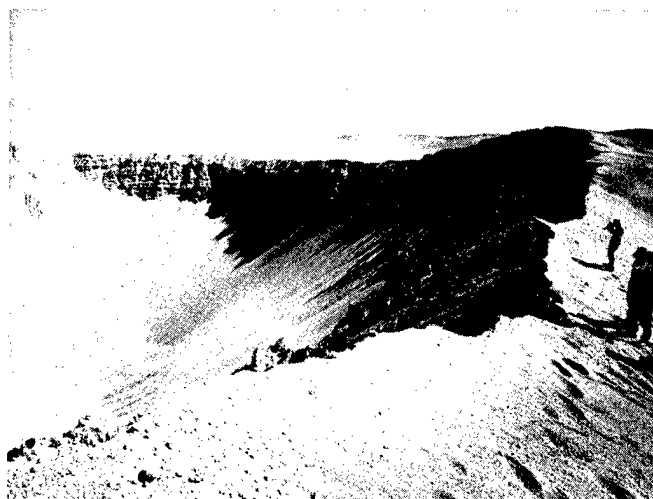
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A STUDY OF SELECTED ROCK EXCAVATIONS AS RELATED TO LARGE NUCLEAR CRATERS

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ENGINEERING PROPERTIES OF NUCLEAR CRATERS
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PREFACE

This empirical study of high rock excavations was conducted for the U. S. Army Engineer Nuclear Cratering Group (NCG) by the U. S. Army Engineer Waterways Experiment Station (WES). During the early stage of the study, the emphasis was on a literature review and a collection of descriptions of example slopes. Mr. F. C. Girucky of the Embankment and Foundation Branch, Soils Division, WES, was responsible for this early stage. Later, Mr. B. N. MacIver, Embankment and Foundation Branch, WES, continued the study by canvassing state and private civil and mining organizations for unpublished data.

This report, prepared by Dr. R. J. Lutton, Geology Branch, WES, and Capt. R. J. Kley, NCG, is a condensation of data previously collected, and data generated by a continuing effort.

The generous response of mining companies and civil works agencies is gratefully acknowledged. A list of these contributing organizations is presented in Appendix B, Part I-B.

ABSTRACT

Analogies between nuclear and conventional excavations are developed from a tabulation of data from 153 mine, quarry, roadway, and dam excavations. The following factors were used as the basis for tabulation of conventional excavation data: purpose, location, precipitation, temperature, ground water level, lithology, mass strength, structural pattern, slope plan, slope profile, depth of excavation, slope height, average inclination, and stability.

It was found that average slope inclination tends to be greatest for hard material and for material lacking a well-developed structure, and that inclination tends to decrease with increasing slope height for excavated slopes reported to be stable.

The authors conclude that good analogies are to be found in shape, slope height, depth of excavation, and slope inclination. Loadings of waste material at the rim of some open pit mines may be analogous to ejecta on the lips of nuclear craters. Rubble zones found in some open pit mines may be analogous to the fallback zones of nuclear craters. Differences between preshot and postshot characteristics of cratered media must be appreciated in evaluating analogies between features of conventional excavations and preshot features of nuclear excavation sites.

Brief descriptions of 21 selected excavations are presented in Appendix A. The results of this study serve as an aid to judgment of nuclear crater slope stability. Potential subjects for further study are identified.

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CHAPTER 1

INTRODUCTION

1.1 GENERAL

1.1.1 Purposes. The purposes of this report are to: (1) present selected data on the geology, geometry, and stability of excavated rock slopes; (2) identify and evaluate possible analogies between significant features of conventionally excavated slopes and those of proposed nuclear craters; (3) serve as an aid to judgment in evaluating the stability of proposed nuclear excavations; and (4) provide a basis for future studies of excavated rock slopes which exhibit specific analogies to nuclear craters.

1.1.2 Scope. This report includes the following:

- Chapter 1 - A general description of large (>100 kt) nuclear craters in rock
- Chapter 2 - A tabulated summary of data concerning the geometry, geology, and stability of conventionally excavated rock slopes
- Chapter 3 - A comparison of certain features of conventional and nuclear excavations
- Chapter 4 - An indication of how quantitative relationships such as inclinations of stable and unstable slopes with respect to slope height, mass strength, and structural pattern may be established from this and future related studies
- Chapter 5 - Conclusions
- Appendix A - Brief descriptions of specific excavations which appear to show some analogy to nuclear craters
- Appendix B - A bibliography of published and unpublished sources of relevant data

1.1.3 Application of Results. Using this report as a guide, additional comparisons can be made between conventionally excavated rock slopes and specific nuclear crater slopes with regard to geology, geometry, and environment. This report also provides a basis for judgment in evaluating the stability of proposed nuclear excavations, and a point of departure for future studies relating conventional and nuclear excavations.

1.2 BACKGROUND

Since 1962 the Atomic Energy Commission (AEC) and the U. S. Army Corps of Engineers have been engaged in a joint research program to develop nuclear excavation technology. Under the agreement for this program, the AEC is primarily responsible for nuclear device development, execution of nuclear cratering experiments, and

development of scaling methods for predicting the size and shape of nuclear craters. The U. S. Army Corps of Engineers is primarily responsible for execution of a corollary chemical high explosive cratering program and, in addition, development of the requisite data on engineering and construction problems associated with nuclear excavation. The ultimate objective of the U. S. Army Corps of Engineers in the nuclear excavation research program is to develop a capability to employ nuclear explosives as a construction tool on various national and international public works projects.

As part of this joint research program, a series of studies of the engineering properties of explosively excavated craters is currently being conducted by the U. S. Army Engineer Nuclear Cratering Group (NCG), together with its prime research contractor, the U. S. Army Engineer Waterways Experiment Station (WES). The major purpose of these studies is to investigate, through theoretical, experimental, and empirical research, those geological and physical characteristics which govern stability of nuclear crater slopes.

Empirical studies are limited by the small number and limited size of existing nuclear craters. Certain analogies may exist, however, between the slopes of proposed nuclear excavations (Figure 1.1) and conventionally excavated rock slopes (Figure 1.2). This study of conventionally excavated slopes identifies characteristics which may be analogous to those of nuclear crater slopes.

1.3 NUCLEAR EXCAVATIONS IN ROCK

Proposed nuclear excavations for roads, canals, and harbors require nuclear explosives ranging in equivalent yield up to 10 Mt* of TNT. Many of these proposed projects involve the simultaneous detonation of several nuclear devices in a row to produce a linear excavation, as illustrated in Figure 1.1.

The available data on characteristics of crater excavations in hard dry rock (basalt and rhyolite) have been obtained by field investigations at the Danny Boy site (References 271, 280, 285) and the Pre-Schooner (Reference 274), Dugout (Reference 273, and Pre-Schooner II (Reference 219) explosive craters. Investigations at several single-charge nuclear craters in desert alluvium (Reference 289) and from high explosive single- and row-charge craters in wet clay shale (References 253, 261) supplement the hard rock crater data. Consideration of all data has led to the conclusion that large nuclear craters in basalt will be similar in shape to the Danny Boy crater, with slope inclinations of approximately 35°. The slope angle may be significantly different in other media.

Figure 1.3 shows an idealized cross section of a single-charge nuclear crater in basalt. Table 1.1 gives approximate row-crater dimensions resulting from the detonation of a linear row of four or more devices of 100 kt to 10 Mt yield at optimum depth of burst. These dimensions were calculated on the basis of Danny Boy crater dimensions enhanced 10 percent, using 1/3.4 scaling. Limited high explosive row-charge cratering

*One kiloton (kt) equals one thousand tons. One megaton (Mt) equals one million tons.

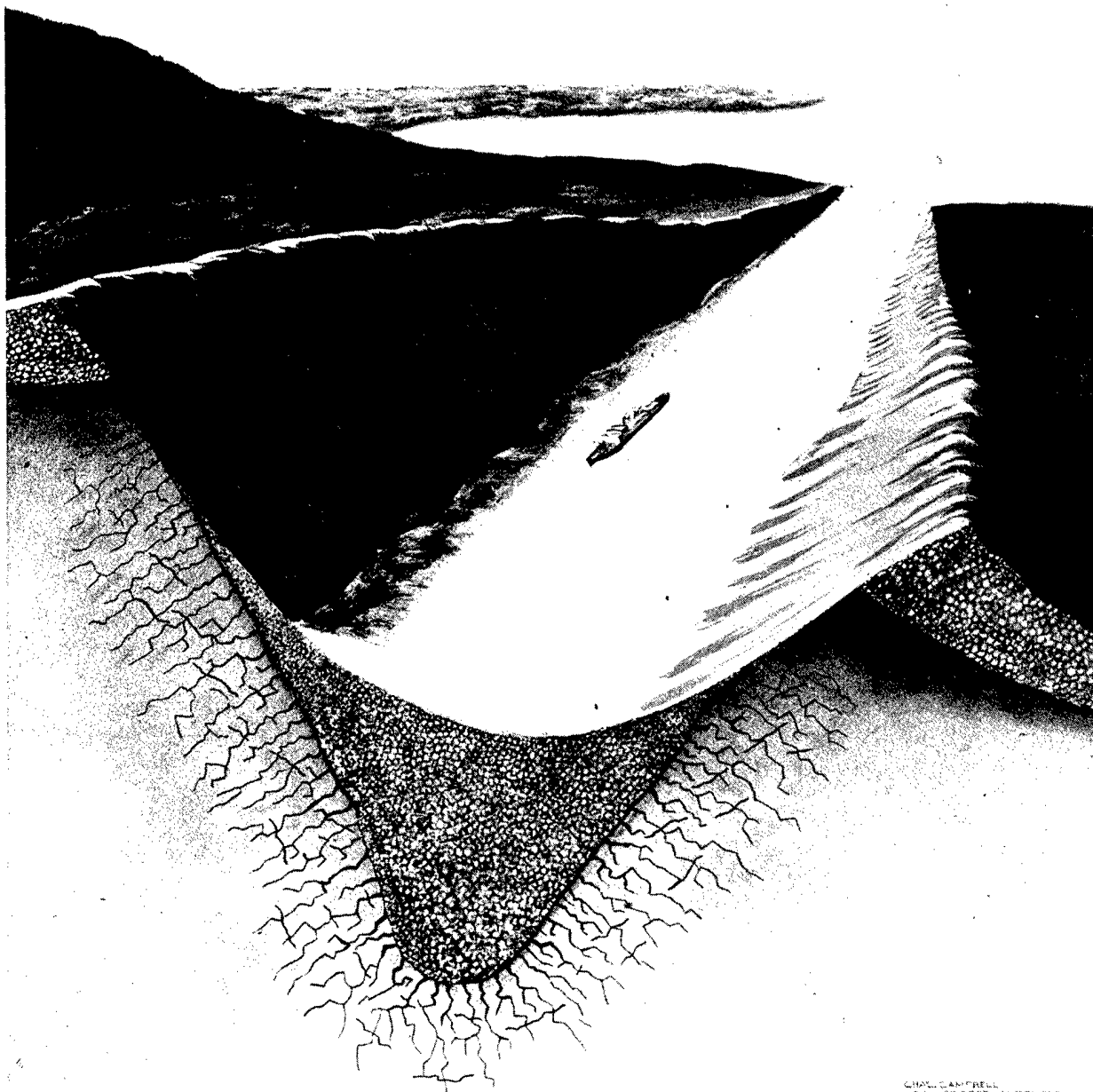


Figure 1.1 Conceptual drawing of nuclear-excavated channel.



Figure 1.2 Kiruna Pit; oblique aerial view looking southward.
(Contributed by Luossavaara-Kiirunavaara Aktiebolag)

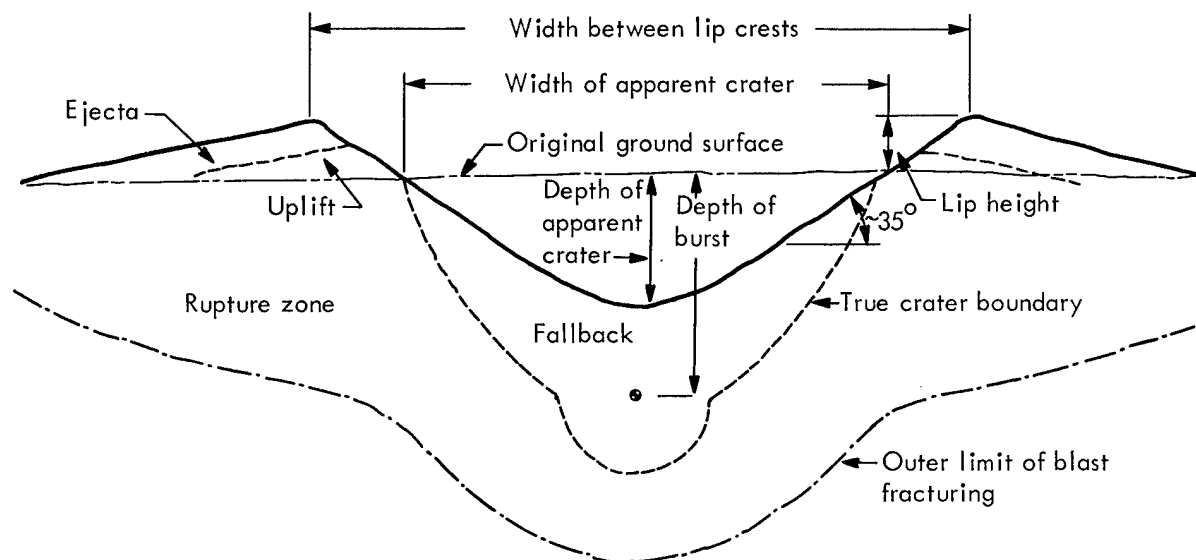


Figure 1.3 Idealized cross section of a single-charge nuclear crater in basalt.

TABLE 1.1 CALCULATED DIMENSIONS OF ROW CHARGE CRATERS*

Individual Charge Yield	Width of Apparent Crater	Depth of Apparent Crater	Design Depth of Burst	Height of Apparent Lip	Width Between Lip Crests
	feet	feet	feet	feet	feet
100 kt	1170	341	550	268	1952
200 kt	1436	417	676	335	2390
500 kt	1876	548	882	440	3134
1 Mt	2300	670	1082	538	3844
2 Mt	2820	820	1370	659	4710
5 Mt	3700	1076	1739	852	6330
10 Mt	4530	1322	2132	1055	7560

* In hard dry rock (basalt), calculated from Danny Boy observed dimensions using 1/3.4 scaling and 10 percent enhancement.

NOTE: Factors for converting British units of measurement to metric units are presented in Table 1.2.

TABLE 1.2 CONVERSION FACTORS, BRITISH TO METRIC UNITS OF MEASUREMENT

British units of measurement used in this report can be converted to metric units as indicated below.

Multiply	By	To Obtain
inches	2.54	centimeters
feet	0.3048	meters
miles	1.609344	kilometers

experience to date indicates that the 10-percent enhancement is a reasonable assumption when converting from single-charge to row-charge dimensions. The scaling relationship is based upon all available experience.

As shown in Figure 1.3, an explosion crater in basalt consists of three zones, as follows:

- Ejecta Zone — Consisting of broken and disarranged material thrown outward from the crater and deposited near the crater rim
- Fallback Zone — Consisting of broken and disarranged material that has come to rest in the crater
- Rupture Zone — Consisting of blast-fractured rock in which relative particle orientation has remained essentially unchanged.

Disturbed rock in the ejecta and fallback zones is bulked from 1.4 to 1.6 times its original volume. Volume increases of lesser degree are found in the rupture zone. Blast fractures increased effective (fracture) porosity from an initial two percent to as much as 25 percent in the rupture zone of the Dugout row crater in basalt (Reference 274).

CHAPTER 2

CONVENTIONALLY EXCAVATED ROCK SLOPES

Hundreds of deep open excavations have been made in rock to satisfy the growing demand for mineral products and the expanding need for large public works. These excavations provide an abundant source of data on slope stability under diverse conditions.

Data for 293 conventionally excavated rock slopes at 153 localities, as shown in Figures 2.1 and 2.2 have been tabulated in Table 2.1 with respect to several significant parameters discussed in the following sections of Chapter 2. Parameters selected were those most related to slope stability. Brief descriptions of selected examples are presented in Appendix A.

2.1 PURPOSE OF ROCK EXCAVATIONS

Man-made excavations may be subdivided into two broad categories: (1) mining engineering (mining), and (2) civil engineering (civil) excavations. Mining excavations include rough stone quarries, dimension stone quarries, industrial rock and mineral quarries, metal mines, and mineral fuel mines. Civil excavations include highway and railroad cuts, canal cuts, and large structural foundation cuts.

Because even the largest scale mining operations are relatively short lived as compared to the average civil excavation, they probably best represent the maximum slope inclination relative to slope height for any given conditions. More than one-half of the slopes cited in this report are in mines.

2.2 LOCATION OF ROCK EXCAVATIONS

Most major civil excavations are located in the industrialized nations of the world. Many mines and quarries are located in or near the highly industrialized nations which provide markets for their products. Because of complex geological circumstances, most major mining excavations are located in mountainous regions. The localities selected for consideration in this report are, therefore, biased in terms of geographical distribution to industrialized or mountainous regions, as illustrated in Figures 2.1 and 2.2. This geographical distribution has been further influenced by the fact that data available in foreign language publications have not been extensively reviewed. The majority of the selected localities, therefore, lie within the English-speaking nations; the greatest concentration of these localities is in the western United States.

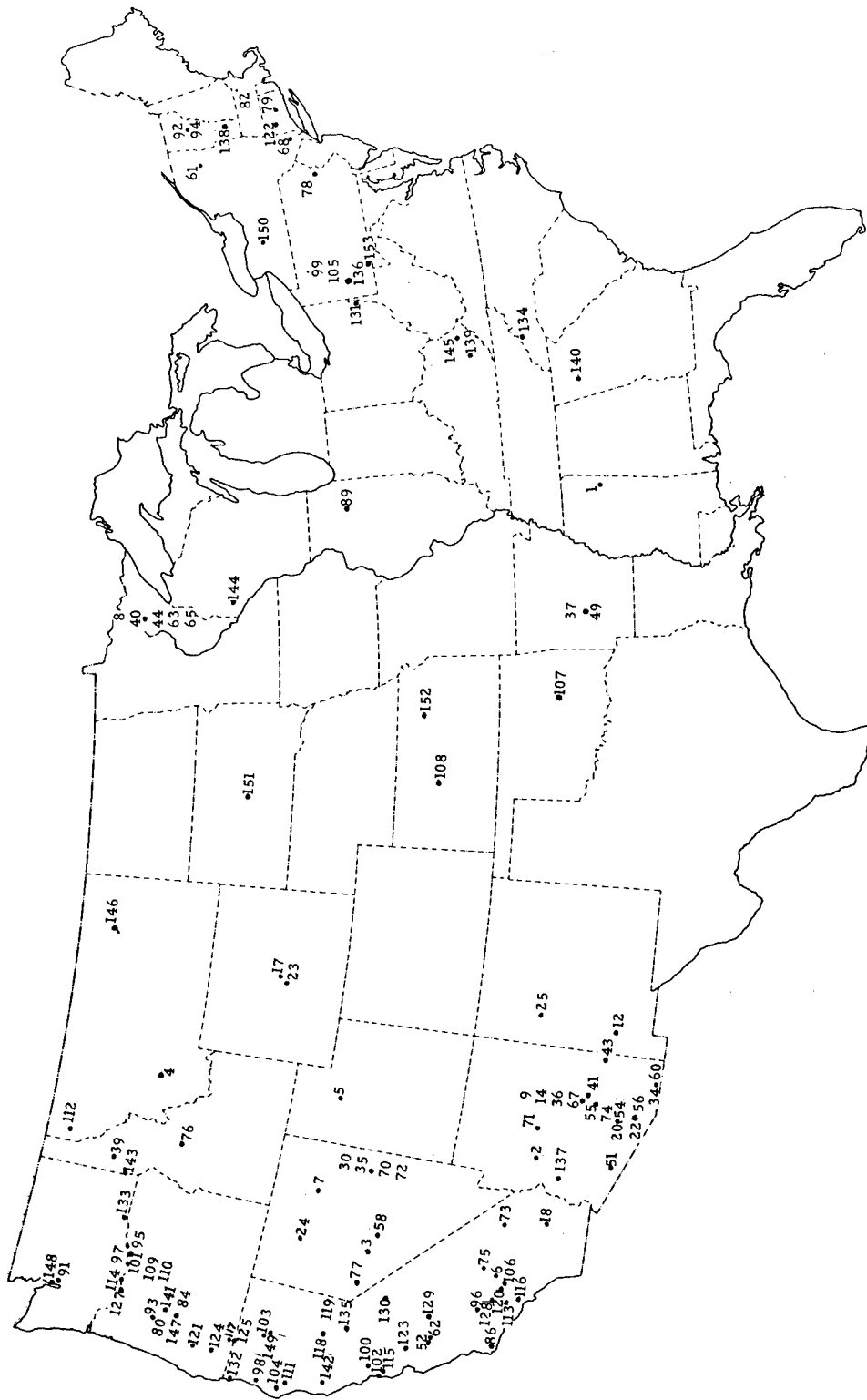


Figure 2.1 Map of example localities within coterminous United States.

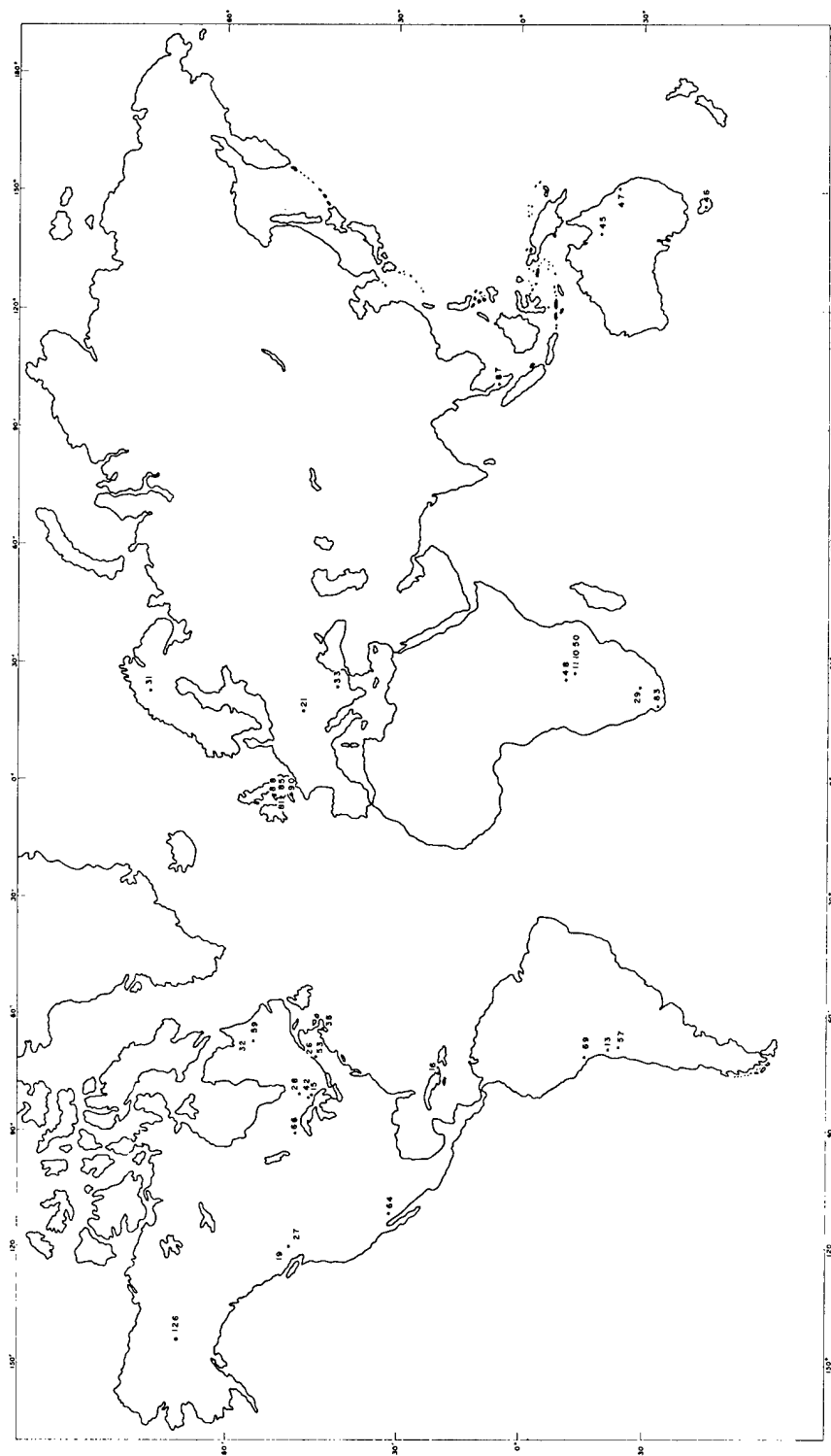


Figure 2.2 Map of example localities outside coterminous United States.

11	Chingola	Chingola, Zambia	Copper	51	68	74	59	Low	Sandstone and dolomite	H	O			240	240	45
12	Chino	Santa Rita, N.M.	Copper	16	56	88	24	Low	Porphyry	H	P	L	CC	550	500	20
13	Chuquicamata	Chuquicamata, Chile	Copper	1	61	69	54	Low	Granodiorite	H	P	L	L	400	400	38
14	Copper Cities	Miami, Ariz.	Copper	19	64	96	33	Low	Quartz monzonite	H	P	L	L	275	300	25
15	Creighton	Sudbury, Ontario	Copper and nickel	40	39	65	9	High	Brecciated norite	H	P	CC	L†	160		70
16	Cuban Mining Co.	Cristo, Oriente, Cuba	Manganese	40	78	83	73	High	Tuff and limestone	S	O			-	-	40
17	Day-Loma	Jeffrey City, Wyo.	Uranium	9	43	89	0	High	Sandstone, and conglomerate	S	O			300	240	61 ^c
18	Eagle Mountain	Riverside Co., Cal.	Iron	3	73	105	43	Low	Quartzite	H	O	L		-	300	20
19	East Jersey	Ashcroft, British Columbia	Copper	10	47	70	23	High	Volcanic rocks	H	P	CC	L	-	400	55
20	El Tiro	Silver Bell, Ariz.	Copper	11	67	100	35	Low	Monzonite and porphyry	H	P	CC		400	450	45
21	Erzberg	Eisenerz, Austria	Iron	33	47	66	25	Low	Limestone	H	P	L	L	270	400	45
22	Esperanza	Twin Buttes, Ariz.	Copper	11	67	100	35	Low	Monzonite, and quartz latite	H	P	L	L	240	310	44
23	Frazier-Lamac	Jeffrey City, Wyo.	Uranium	9	43	89	0	High	Sandstone	S	O			600	2,200	42
24	Getchell	Golconda, Nev.	Gold	6	48	92	16	Low	Argillite, and limestone, and clay	H	P	L	L†	300	318	49
25	Jackpile	Grants, N.M.	Uranium	9	52	89	17	High	Shale and sandstone	H	O	L		300	290	62
26	Jeffrey	Thetford, Quebec	Asbestos	38	44	71	15	High	Phyllite, and quartzite, and ultramafics	H	P	CC	CC	450	450	15
27	Jersey	Ashcroft, British Columbia	Copper	10	47	70	23	High	Volcanic rocks	H	P	CC	L	-	200	40
28	Kam Kotia Porcupine	Boston, Ontario	Copper and zinc	33	34	63	0	High	Greenstone	H	P	L		-	180	60
29	Kimberley	Kimberley, So. Africa	Diamonds	16	65	77	51	High	Shale, serpentine	H	P	CC	L†	600	600	56

TABLE 2.1 (Continued)

Locality ¹ Number	Name of Excavation	Location	Product or Purpose	Annual Precipi- tation	Relative Ground Water				Mass ² Strength	Struc- tural ³ Pattern	Slope ⁴ Plan	Slope ⁵ Profile	Exca- vation Depth	Slope		Average Inclination ⁶	
					Mean	High	Low	Low						feet	degrees		
																	Mines (Continued)
30	<u>Kimbley</u>	Ruth, Nev.	Copper	8	44	86	9	Low	Porphyry and shale	CC	L	-	500	60			
										L	CC	350	430	45			
										CC	L	450	500	46			
										CC	L	450	425	43			
										CC	L	450	425	43			
										CC	CV	450	280	55			
										CC	CC	450	260	40			
										CC	L	450	475	44			
										CC	CC	450	250	35			
										CC	L	450	410	40			
										CC	L	450	390	25			
										CC	CC	450	310	25			
										CC	CV	450	410	50			
										CC	L	450	480	55			
										CC	L	450	500	55			
										CC	L	450	480	55			
										CC	L	450	390	57			
31	<u>Kiruna</u>	Kiruna, Sweden	Iron	15	29	56	7	-	Syenite and magnetite	L	L	-	550	45			
										L	L	460	400	54			
										L	L	480	640	44			
										L	L	500	520	50			
										L	L	385	370	54			
										L	L	450	600	44			
32	Knob Lake	Schefferville, Quebec	Iron	30	24	55	-8	High	Slate and quartzite	L	L	310	310	31			
										L	L	310	210	35 ^a			
												170	170	32			
												170	70	52 ^a			
33	Kremikovtzi	Sofia, Bulgaria	Iron	25	51	70	30	High	Limestone	P	P	270	270	42			
34	Lavender	Bisbee, Ariz.	Copper	18	61	90	33	Low	Porphyry	P	P	500	400	30			
35	Liberty	Ruth, Nev.	Copper	8	44	86	9	Low	Porphyry and limestone	P	P	900	850	35 ^a			
										CC	L	300	400	40			
										CC	L	200	220	45			
										CC	CC	-	490	35			
										L	L	-	550	23 ^b			
										L	L	-	650	38			
										L	CV	-	840	29			
36	<u>Live Oak</u>	Miami, Ariz.	Copper	19	64	96	33	Low	Porphyry	P	P	300	650	37 ^a			
										CC	L	-	500	36 ^a			
										CV	L	-	650	34 ^a			
										CC	L	-	175	23			
										L	L	-	260	15			
										L	CV	-	200	45			
37	Magcobar	Magnet, Ark.	Barite	52	63	93	32	-	Shale and sandstone	O	L	160	190	40			
											L	200	230	60			
38	Magnet Cove	Walton, Nova Scotia	Barite, lead, and zinc	55	45	64	27	High	Limestone breccia and siltstone	O	CC	350	350	25			
											L	350	350	45			

39	Miclasil	Bovill, Ida.	Mica, clay, and silica	22	47	82	21	High	Granite and metamorphic rocks	S	D	L	-	105	60
40	<u>Missabe Mountain</u>	Virginia, Minn.	Iron	27	39	80	-4	High	Chert and slate	S	O	L	L [†]	425	65
41	Mission	Tucson, Ariz.	Copper	16	62	99	29	Low	Porphyry and sedimentary rocks	H	P	L	L [†]	410	55
42	Moose Mountain	Capreol, Ontario	Iron	40	39	65	9	High	Taconite and volcanics	H	O	L	L	560	45
43	Morenci	Morenci, Ariz.	Copper	12	66	99	33	Low	Porphyry	H	P	CC CC L	700	900	20
												CC L	-	690	19
												CC	-	450	22
												CC	-	490	25
												CC	-	950	20
												CC	-	1,060	19
												CC	-	1,000	19
44	Morton	Hibbing, Minn.	Iron	27	39	80	-4	High	Taconite, and hematite, and slate	S	O		250	250	70 ^c
45	Mount Isa	Mount Isa, Australia	Copper and lead	18	72	85	60	Low	Shale and quartzite	H	O	L	-	460	35
46	Mount Lyell	Queenstown, Tasmania	Copper	90	55	65	45	High	Schist and conglomerate	H	O	L	-	400	45
47	Mount Morgan	Queensland, Australia	Copper	50	68	78	62	High	Granite and metamorphic rocks	H	O	CC	L	600	40
48	Musonoi	Kolwezi, Katanga, Congo	Copper	53	69	75	63	High	Sandstone	H	O		100	500	35
49	National Lead	Magnet, Ark.	Barite	52	63	93	32	-	Shale and sandstone	H	O	L	CV	190	40
50	Nchanga	Chingola, Zambia	Copper	51	68	74	59	Low	Sandstone and dolomite	H	O	L	L	450	30
51	<u>New Cornelia</u>	Ajo, Ariz.	Copper	9	71	103	40	Low	Quartz monzo- nite and rhyolite	H	P	L	CC	500	23
													300	460	37
52	New Idria	Idria, Cal.	Mercury	15	63	90	35	Low	Sandstone and serpentine	H	P	CC	CC [†]	180	40
53	Normandie	Black Lake, Quebec	Asbestos	38	44	71	15	High	Ultramafics	H	D	CC	L	450	45

TABLE 2.1 (Continued)

Locality ¹ Number	Name of Excavation	Location	Product or Purpose	Annual Precipitation	Temperature			Relative Ground Water Level	Materials	Mass ² Strength	Struc- tural ³ Pattern	Slope ⁴		Exca- vation Depth	Slope Height	Average Inclination ⁶
					Mean	High	Low					Plan	Profile			
Mines (Continued)																
54	Oxide	Silver Bell, Ariz.	Copper	11	67	100	35	Low	Monzonite and porphyry	H	P		L	350	250	45
												CV	L	190	280	50
55	<u>Pearl Handle</u>	Ray, Ariz.	Copper	19	64	96	33	Low	Schist	H	O	CC	CC	200	450	37a
												CC	L	-	300	45
												CC	L	-	350	24
												CC	L	-	500	37
56	Pima	Tuscon, Ariz.	Copper	11	67	100	35	Low	Prophyry and sedimentary rocks	H	P			400	400	41
57	Quinta Tercera	Mantos Blancos, Chile	Copper	1	61	69	54	Low	Rhyolite and andesite	H	P		L	-	350	26a
58	Round Mountain	Round Mountain, Nev.	Gold	6	49	98	16		Sand and gravel	S	D			120	120	70
59	Ruth Lake	Schefferville, Quebec	Iron	30	24	55	-8	High	Schist	S	O			445	445	51 ^a
60	Sacramento Hill	Bisbee, Ariz.	Copper	18	61	90	33	Low	Quartzite, limestone, and granite	H	P			-	425	45
61	<u>Sanford Lake</u>	Tahawus, N. Y.	Titanium	37	41	77	5	High	Gabbro, anorthosite, and magnetite	H	P	CC	L	700	700	50
												L	CV	-	380	39
												CC	CV	-	420	51
												L	L	-	220	24
												CV	L	-	610	46
												CC	L	-	205	43
												CC	L	-	245	37
62	Santa Cruz	Coalinga, Cal.	Asbestos	15	63	90	35	Low	Serpentine	S	D		L	-	100	45 ^c
63	Schley	Virginia, Minn.	Iron	27	39	80	-4	High	Slate and taconite	S	O			350	350	30
														75	75	34 ^a
64	Sonora Hill	Cananea, Sonora, Mexico	Copper	25	59	76	43		Porphyry and pyroclastics	H	P	CV	L	-	420	50 ^a
												L	L	-	315	62
												L	L	-	455	60
												L	L	-	490	58
65	South Agnew	Hibbing, Minn.	Iron	27	39	80	-4	High	Slate and taconite	S	O			200	200	75
66	<u>Steep Rock Lake District</u>	Atikokan, Ontario	Iron	27	39	80	-4	High	Limestone, hematite, and pyroclastics	H	O		L	215	215	51 ^a
														420	550	55
67	<u>Thornton</u>	Miami, Ariz.	Copper	19	64	96	33	Low	Schist and porphyry	H	P		CC	300	750	35 ^a
												CV	L†	-	600	30 ^a
												CC	L†	-	200	35
												CC	L†	-	450	46
												L	L†	-	235	50
												L	L	-	275	21

68	Tilly Foster	Tilly Foster, N. Y.	Iron	51	51	83	22	High	Magnetite and gneiss	H	O	CC	CV†	-	300	64
69	Toquepala	Toquepala, Peru	Copper	1	62	69	54	-	Altered volcanics	H	D	CC	L	-	650 1,100	30 20
70	Tripp	Ruth, Nev.	Copper	8	44	86	9	Low	Limestone and porphyry	H	P		L	450 450	450 350	35 40
71	United Verde	Jerome, Ariz.	Copper	17	60	90	32	Low	Diorite, schist, and porphyry	H	P	CC	CV	540 500	1,000 700	45 63a
72	Veteran	Ruth, Nev.	Copper	8	44	86	9	Low	Limestone and porphyry	H	P	CC	L	450	450	35
73	Vulcan	Kelso, Cal.	Iron	3	73	108	41	Low	Quartz monzo- nite, limestone, and magnetite	H	P			100	300	45
74	West	Ray, Ariz.	Copper	19	64	96	33	Low	Schist	H	O			300	500	45a
75	Yellow Aster	Randsberg, Cal.	Gold	6	63	99	35	Low	Granite and schist	S	P	CC	L†	300	450	45a
76	Yellow Pine	Stibnite, Ida.	Antimony	26	40	84	6	High	Granodiorite	H	O			170	330	37
77	Yerington	Yerington, Nev.	Copper	5	50	93	15	Low	Porphyry and granodiorite	H	D		L	400	400	45a
Quarries																
78	Alpha	Wind Gap, Pa.	Slate	48	50	86	18	High	Slate	H	O	L	CV†	-	260	90
79	Brownstone	Portland, Conn.	Dimension	48	50	84	20	High	Sandstone	H	O	L	L†	160	160	75
80	Cougar	Blue River, Ore.	Rockfill	71	50	85	27	High	Andesite	H	P	L	L	250	400	58
81	Dorothea	England	Slate	27	49	60	38	High	Slate	H	O			-	450	45
82	Fletcher	West Chelmsford, Mass.	Dimension stone	42	50	83	17	High	Granite	H	D	L	L	225	215	45
83	Glencairn	South Africa	Aggregate	25	63	71	55	-		H	O	CC	L	-	400	70
84	Hills Creek	Oakridge, Ore.	Rockfill	44	53	85	30	High	Andesite	H	D	CC	L	150	200	52a
85	Kirkby	Kirkby, Lanca- shire, England	Slate	31	49	60	38	High	Slate	H	O		CC	-	400	50
86	Lompoc	Lompoc, Cal.	Diatomite	13	57	71	39	-	Diatomaceous earth	S	P	L	L	-	170	85
87	Lunchoo	Malaya, Malaysia	Crushed rock	85	84	87	80	High	Granite	H	D	L	L†	-	250	75
88	Minera	Wales	Limestone	27	49	60	38	High	Limestone	H	O		L	220	220	90

TABLE 2.1 (Continued)

Locality ¹ Number	Name of Excavation	Location	Product or Purpose	Annual Precipitation	Temperature		Relative Ground Water Level	Materials	Mass ² Strength	Struc- tural ³ Pattern	Slope ⁴ Plan	Slope ⁵ Profile	Exca- vation Depth	Slope	Average
					Mean	High								feet	degrees
Quarries (Continued)															
89	Ottawa	Ottawa, Ill.	Sandstone												
90	Penrhyn	Penrhyn, Corn- wall, England	Slate	39	52	61	42	High	H	O	L	L	125	125	90
91	Renton	Renton, Wash.	Shale	34	52	74	36	High	S	O		L	150	270	45
92	Rock of Ages	Barre, Vt.	Dimension stone	34	43	79	8	High	H	D	L	L	275	275	85
93	Shot Pouch	Sweethome, Ore.	Rockfill	62	51	82	29	High	H	O	L	L	150	250	62
94	Wells- Lamson	Barre, Vt.	Dimension stone	34	43	79	8	High	H	D			—	480	85
Roadcuts															
95	Arlington	Arlington, Ore.	Highway	9	54	90	26	High	S	P	L	L [†]	90	211	29
96	Bakersfield	Bakersfield, Cal.	Railroad	10	55	88	28	Low	S	D		L	—	200	45 ^a 30
97	Blalock	Blalock, Ore.	Highway	14	54	87	28	Low	H	P	L	L	120	300	56
98	Blue Lake	Blue Lake, Cal.	Highway	60	52	62	41	High	S	O		L	96	240	39 ^a 22 ^a
99	Brilliant	Pittsburgh, Pa.	Railroad	36	53	84	24	High	H	O			—	200	45 ^a
100	Carquinez	Crockett, Cal.	Highway	24	58	81	38	Low	S	O	L	L	300	350	21
101	Cliffs	Rufus, Ore.	Railroad	14	54	87	28	Low	H	P	L	L [†]	220	180	53
102	Devils Slide	Pacifica, Cal.	Highway	21	57	68	46	High	S	O	L	L [†]	50	1,000	35 ^a
103	Dunsmuir	Dunsmuir, Cal.	Railroad	58	49	87	23	High	S	D	—	—	—	100	47 ^a 33 ^a
104	Dyerville	Weott, Cal.	Highway	65	55	70	40	Low	H	O	L	L	100	480	36
105	East Bow	Pittsburgh, Pa.	Railroad	36	53	84	24	High	H	O			—	200	45
106	Edwards	Edwards, Cal.	Foundation	5	61	99	29	Low	H	P			—	100	90
107	Eufaula	Eufaula, Okla.	Highway	42	62	95	29	High	H	O	L	CC	65	150	24 ^a 31
108	Fontana	Fontana, Kan.	Railroad	35	54	90	18	—	H	O			—	65	33 ^a 13 ^a

109	Helms Creek Rufus, Ore.	Highway	14	54	87	28	Low	Breccia	H	P	L	L [†]	40	538	33
110	John Day River	Highway	14	54	87	28	High	Breccia and basalt	H	P	L	CV	140	200	58
111	Kekawaka	Railroad	65	55	70	40	-	Sandstone and mudstone	S	P			1,700		14a 11b
112	Libby Project	Highway	19	45	87	13	Low	Argillite	H	O			145	205	73
113	Liebre	Highway	12	56	86	34	Low	Shale and sandstone	S	O		L	220	360	27a 19b
114	Meredith Point	Highway	31	50	81	27	High	Andesite	H	O	L	L [†]	150	170	76a
115	Monte Vista	Highway	55	77	91	34	High	Greenstone	S	P		L	60	150	45
116	Mulholland	Highway	15	64	82	46	High	Sandstone and shale	S	O	L	L [†]	320 270	370 240	35a 18
117	North Grade	Highway	20	52	86	30	-	Granite	H	O		L	60	200	31a
118	Oroville	Highway	29	61	97	36	High	Phyllite	H	O	L	L [†]	-	100	53
												L	-	100	34a
												L	-	100	26
119	Oroville	Railroad	29	61	97	36	High	Phyllite	H						
120	Red Rock Canyon	Highway	3	64	103	30		Rhyolite	H	O	L	L	-	120	52
121	Roberts Mountain	Highway	33	54	83	35	High	Shale, sandstone, and schist	H	O	L	L	42	158	52a
122	Route 8	Highway	41	49	82	18	High	Schist and gneiss	H	O	L	L [†]	-	185	71c
123	Route 152	Highway	9	63	101	36	High	Sandstone and shale	H	O	-	-	56	182	40a
													115	300	26a
													130	170	39a
													114	152	32a
													215	205	30a
124	Sexton Mountain	Highway	30	54	90	32	High	Mudstone and greenstone	H	O	L		140	160	30
125	Siskiyou Summit	Highway	20	52	86	30	Low	Breccia and basalt	H	P	L	L	230	230	39
126	Tanana Crossing	Highway	11	25	72	-18	Low	Schist	S	O	L	L [†]	80	100	75a
127	Tanner Creek	Highway	76	52	79	31	High	Agglomerate and talus	H	D	L	L [†]	40	165	29
128	Tehachapi	Channel	10	55	89	28		Granodiorite	H	D			140	170	76
129	Terminus Dam	Highway	20	62	99	33	-	Granite	H			L	72	94	62a
													75	146	31

TABLE 2.1 (Continued)

Locality ¹ Number	Name of Excavation	Location	Product or Purpose	Annual Precipitation	Relative Ground Temperature				Materials	Mass ² Strength	Structural ³ Pattern	Slope ⁴ Plan	Slope ⁵ Profile	Exca- vation Depth	Slope Average ⁶	
					Mean	High	Low	Water Level							feet	degrees
Roadcuts (Continued)																
130	Tioga Pass	Tioga Pass, Cal.	Highway	35	53	90	24	Low	Granite	H	P			97	110	55
131	Tunnel No. 10	Steubenville, Ohio	Railroad	42	52	85	22	High	Sandstone and mudstone	H	O	L	L	-	200	45
132	U. S. 199	Crescent City, Cal.	Highway	90	53	67	40	Low	Shale and sandstone	S	O	L	L	98	416	36 ^a
133	Wallula Gap	Wallula, Wash.	Highway	16	53	89	27	Low	Basalt	H	P			70	300	76
134	Waterville Lake	Haywood Co., N. C.	Highway	46	54	83	27	High	Quartzite and graywacke	H	O	L		80	350	45 ^a
135	Weimar	Weimar, Cal.	Highway	55	77	91	34	Low	Argillite	H	O		L	60	150	45 ^a
136	West End Bypass	Pittsburgh, Pa.	Highway	36	53	84	24	High	Shale and sandstone	H	O	L	L	65	120	45 ^a
137	Alamo	Salome, Ariz.	Intake	4	71	109	36	Low	Gneiss	H	O	CC	L	-	115	68
138	Ball Mountain	Jamaica, Vt.	Spillway	53	40	75	5		Schist	H	O	L	L [†]		180	76
139	Buckhorn	Buckhorn, Ky.	Spillway	46	57	90	28	High	Sandstone and shale	H	O	L	CV	250	300	65
140	Carters	Carters, Ga.	Headrace	51	61	90	33	High	Quartzite and argillite	H	O	L	L	300	300	60
141	Cougar	Blue River, Ore.	Spillway	71	50	85	27	High	Basalt and tuff breccia	H	P	L	L	-	500	57
												CV	L	-	500	50
												CC	L	-	450	59
142	Coyote	Ukiah, Cal.	Spillway	36	59	94	35	Low	Clay, silt, and sand	S	O	L	L	115	265	31 ^a
														115	273	29
143	Dworshak	Lewiston, Ida.	Tunnel	26	51	93	22	High	Gneiss	H	O	L	L	197	337	39 ^a
144	Eau Galle	Spring Valley, Wis.	Abutment	31	45	84	5	High	Dolomite and sandstone	H	O			70	227	45
145	Fishtrap	Pikeville, Ky.	Spillway	46	57	90	26	High	Sandstone, and siltstone, and shale	H	O	L	L	220	400	55
146	Fort Peck	Fort Peck, Mont.	Abutment	11	41	88	-5	High	Indurated clay	S	P			-	65	34 ^a
														-	45	40 ^a
														-	150	18 ^a
														-	65	26 ^a
														-	150	22 ^a

147	Hills Creek	Oakridge, Ore.	Abutment	40	52	82	32	High	Tuff breccia	H	P	CV	L	200	380	41
148	Mayfield	Tacoma, Wash.	Penstock slope	39	52	74	35	High	Sandstone, shale and basalt	H	O			-	170	45
149	McCloud	Shasta Co., Cal.	Spillway	37	50	85	25	Low	Metamorphic rocks, andesite, and conglomerate	H	O	L	L	150	240	40
150	Mt. Morris	Mt. Morris, N. Y.	Abutment	26	48	83	17	High	Shale	S	O			73	293	48
151	Oahe	North Pierre, S. Dak.		17	47	89	7		Marl, shale, and alluvium	S	O		L	70	259	42
152	Tuttle Creek	Manhattan, Kan.		32	55	93	18		Shale and limestone	H	O			-	120	40
153	Youghiogheny	Confluence, Pa.	Spillway	42	49	83	21	High	Sandstone, siltstone, and mudstone	H	O	L	CV	200	310	51

¹See index maps (Figures 2.1, 2.2). Underlining indicates that locality is described in Appendix A.

²Mass strength: hard (H) when blasting is necessary in excavating; soft (s) when little or no blasting is necessary.

³Structural pattern: ordered (O), partially ordered (P), and disordered (D) as explained in text (Section 2.6).

⁴Slope plan: linear (L), concave (CC), or convex (CV).

⁵Slope profile: linear (L), concave (CC), or convex (CV); dagger (†) indicates slope without benches.

⁶Average inclination: superscript (a) indicates unstable slope; superscript (b) indicates inclination after adjustment; superscript (c) indicates instability in excavation, but no necessarily on this slope.

2.3 CLIMATIC CONDITIONS AT EXCAVATION SITES

Climatic conditions are described in Table 2.1 in terms of average annual precipitation (rainfall equivalent) and monthly average temperatures.

The effect of climate upon slope stability is twofold. First, climatic conditions influence the chemical and physical changes which may alter the strength of the rock. Second, climatic conditions influence seasonal variations of the ground water level.

The excavations selected for consideration in this report represent a limited variety of climates. Approximately 49 percent of the 293 slopes are located in semi-arid regions; 38 percent are situated in regions having a temperate climate.

Data concerning excavated rock slope performance in humid tropical and arctic climates are very limited because of the following:

- a. These areas are accessible only with difficulty, which makes them unfavorable for both the discovery and exploitation of mineral deposits.
- b. Most open pit mines in humid tropical areas exploit materials which have been concentrated in residual deposits formed by tropical weathering. Many of these materials are not consolidated, and few of these mines have been excavated to depths in excess of 100 feet.
- c. Humid tropical and arctic areas are generally underdeveloped and lack both the need and the resources for large-scale civil projects. The Panama Canal, which is the outstanding exception to this generalization, has been partly excavated through unconsolidated media.

2.4 GROUND WATER

In Table 2.1 the relative position of the water table is given as high or low.

The presence of ground water in the walls of an excavation may adversely influence slope stability. Instances of instability in excavated rock slopes (including the Bingham Canyon,* Esperanza,* East Jersey,* and Day-Loma* pits) have been attributed at least in part to the presence of excessive ground water.

The adverse influence of ground water upon a slope results from one or more of the following effects:

- a. Increased stress behind the slope due to seepage forces and the weight of pore water or fracture-filling water.
- b. Decreased effective stress behind the slope due to the pressure of pore water or fracture-filling water, resulting in decreased mass strength.
- c. Decreased mass strength as a result of erosion, solution, or hydration by ground water.

The significance of any of these effects is likely to be increased if the level of the water table is high relative to the depth of the excavation.

* Asterisk indicates localities which are described in Appendix A.

2.5 MEDIA AND THEIR PROPERTIES

In Table 2.1 the geological media at each locality have been arbitrarily classified, as follows:

HARD	Must be broken or loosened by blasting (Figure 2.3)
SOFT	Can be worked in place by earthmoving equipment with minimal blasting (Figure 2.4)
ORDERED	Planar structures (faults, joints, bedding planes, etc.) have a simple regular pattern
PARTIALLY ORDERED	Lacks a simple regular pattern such as sedimentary layering but has some preferred structural orientation; excavations having a single major through-going fracture have been placed in this category
DISORDERED	Planar structures are absent or randomly oriented.

The lithology, structure, and physical properties of geological media encountered in conventional excavations vary widely. Dimension stone quarries (Figure 2.3) exploit the strength and durability of massive unweathered rock. Many open pit iron mines are



Figure 2.3 Rock of Ages Quarry; showing near-vertical slopes in hard massive rock.
(Contributed by Rock of Ages Corporation)

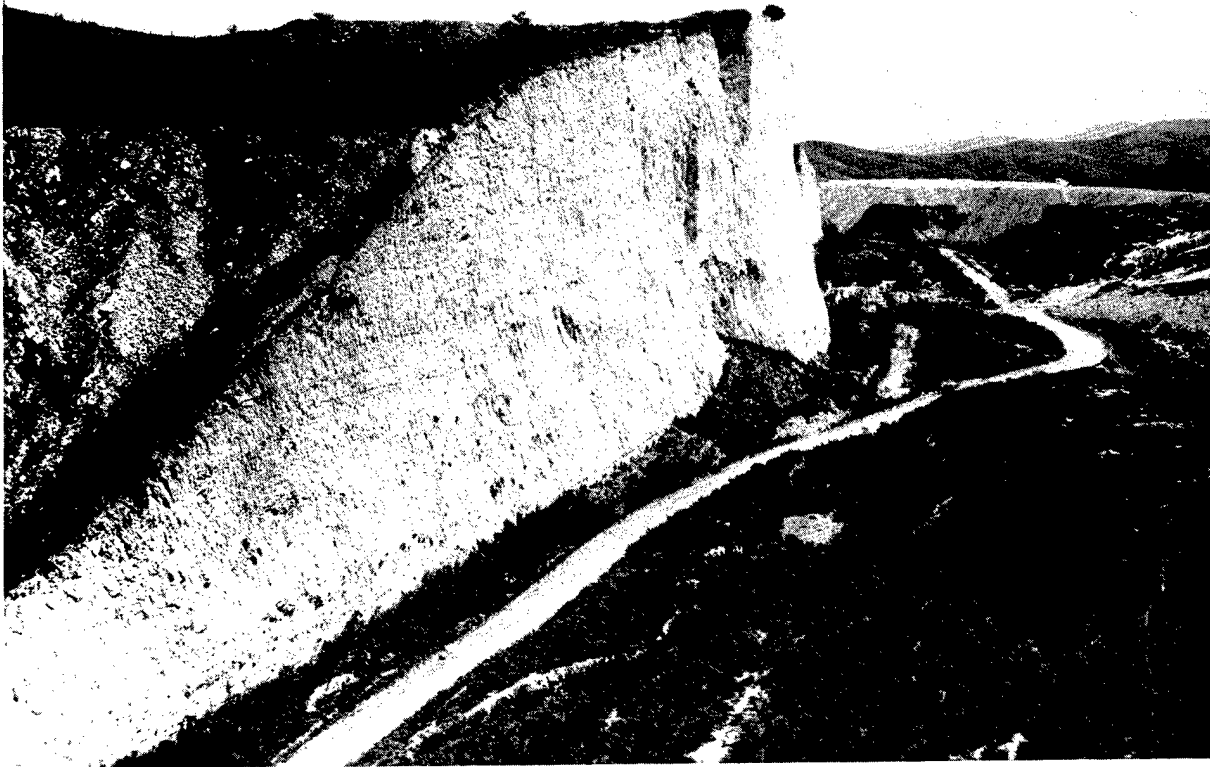


Figure 2.4 Excavated slope near Lompoc Quarry; showing near-vertical 170-foot slope in soft massive material. (Contributed by Johns-Manville Products Corporation)

in soft weathered materials which can be excavated with mechanical earthmoving equipment. Most of the slopes listed in Table 2.1 are in hard rock overlain by a layer of relatively soft and porous weathered material. Civil excavations are generally shallower than major open pit mine excavations and encounter a relatively greater proportion of weathered materials.

Because the processes of ore formation often accompany folding and faulting, mines are frequently located in areas having a complex geological structure. In contrast, most of the civil excavations considered in this report are located in rocks characterized by a relatively simple structure.

Geological structures such as faults and bedding planes which dip toward an excavation are reported to have been a major cause of instability at several localities including the Bingham Canyon Mine,* East Jersey Pit,* Live Oak and Thornton Pits,* and the Mulholland Cut.* The stability problems attributed to such adversely oriented structures have been accentuated in some cases (e. g., at the Bingham Canyon Mine* and the East Jersey Pit*) by the effects of ground water, as described in Section 2.4.

The location of a civil engineering project can sometimes be adjusted to take advantage of favorable geological conditions and to avoid unfavorable ones; but the location of a mine is fixed by that of the ore body.

2.6 SLOPE CONFIGURATION IN PLAN AND PROFILE

In Table 2.1 the configuration of each slope is described as linear, concave, or convex both in plan and profile.

Many of the major open pit mines tabulated in Table 2.1 are approximately circular (concave) in plan. Several of the mines and civil excavations, however, including the Jackpile Mine (Figure 2.5) and the Kiruna Mine* (Figure 1.2) are elongate or linear. The difference between elongate and circular excavations may bear upon problems of slope performance. To the extent that this is true, elongate and linear excavations are most pertinent to slope stability in nuclear-excavated cuts.

Profile configuration may also influence slope stability. Experience in deep granite quarries (Reference 140) has shown that stresses tend to be concentrated in sharply angular reentrants or bench edges. The profiles of most open pit mines and



Figure 2.5 Jackpile Mine; showing linear trend of pit walls.
(Contributed by The Anaconda Company)

major civil excavations are modified by benches. Slopes which are reported to be unbenched are identified in Table 2.1.

2.7 HEIGHT OF SLOPE AND DEPTH OF EXCAVATION

Table 2.1 lists both the height of slopes and the depth of excavation, defined as the difference in elevation between a point at the toe of the slope and the projection of that point on the original ground surface (Figure 2.6). The depth of excavation may equal

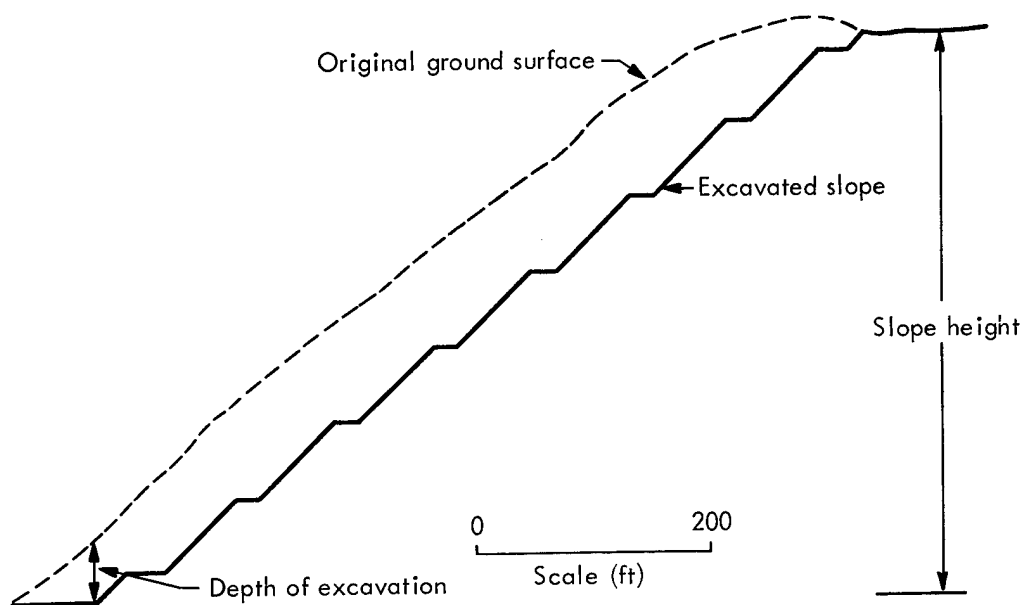


Figure 2.6 Dyerville Cut; representative profile showing contrast between slope height and depth of excavation. (Adapted from reference 121, Smith and Cedergren, 1962)

slope height in pits developed in level terrain, or it may be greater or less than the slope height in the case of excavations into hilltops or hillsides, respectively.

Open pit mining is currently being carried on at depths of over 500 feet in many localities; designed depths in excess of 1,000 feet are anticipated in several of these mines. Current research to improve excavation techniques and develop reliable slope stability criteria are expected to result in successful mining operations at even greater depths in the future.

Slopes considered in this report range up to 2,200 feet in height. In general, slopes with heights of less than 100 feet were arbitrarily eliminated from consideration. A few slopes of lesser height, however, are included for reasons of special geological or engineering interest.

In evaluating and comparing the stability of excavated slopes in various geological media, one must consider not only the height of the slope, but also the thickness of rock removed. Hillside cuts which remove only a layer of material parallel to the original surface (Figure 2.7) may be less relevant to the problems of slope stability than those

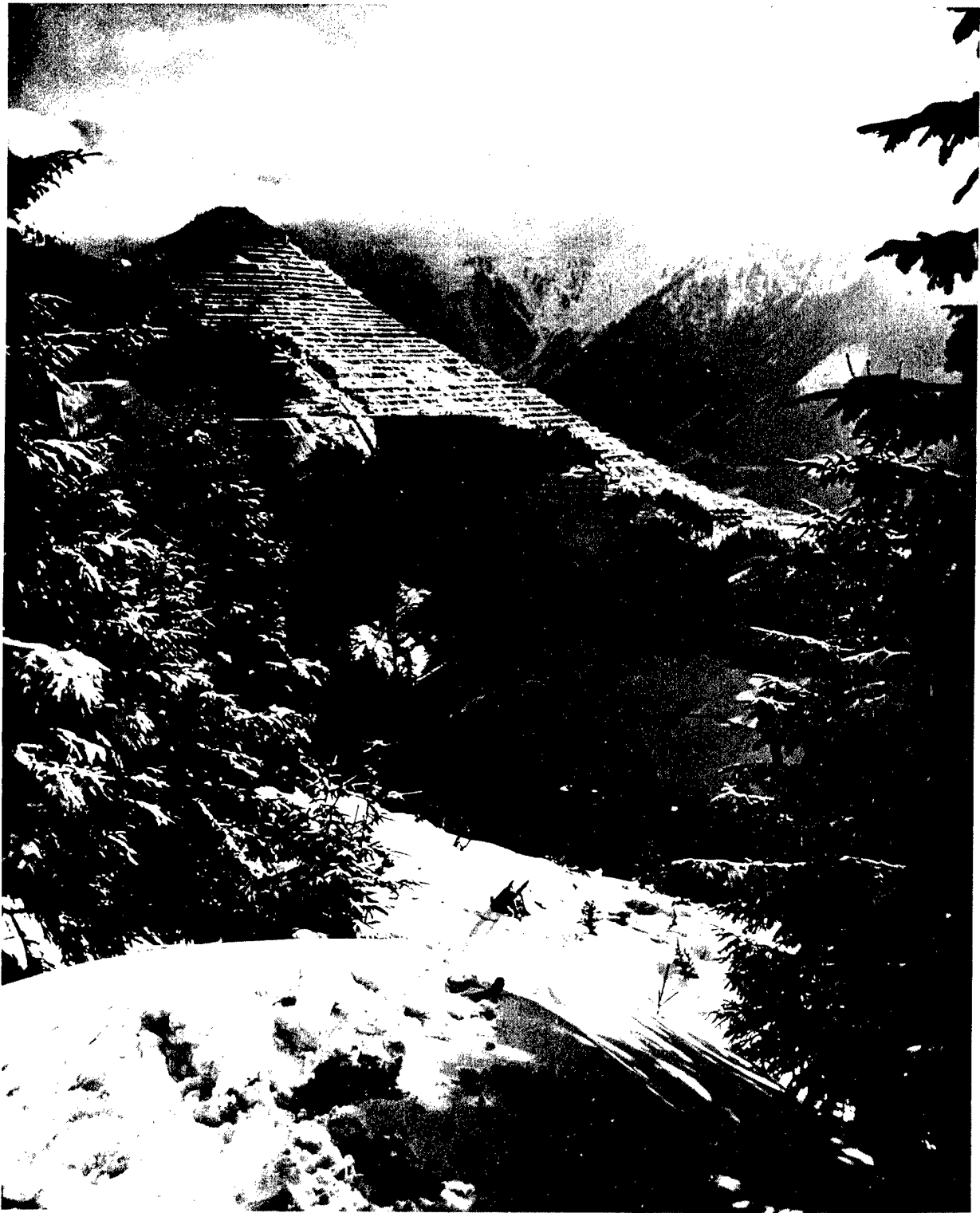


Figure 2.7 Erzberg; a high rock slope produced without deep excavation.
(Contributed by Oesterreichisch-Alpine Montangesellschaft)

pits in level terrain (Figure 2.8), or those such as the Mulholland Cut* which pass through a hill mass.

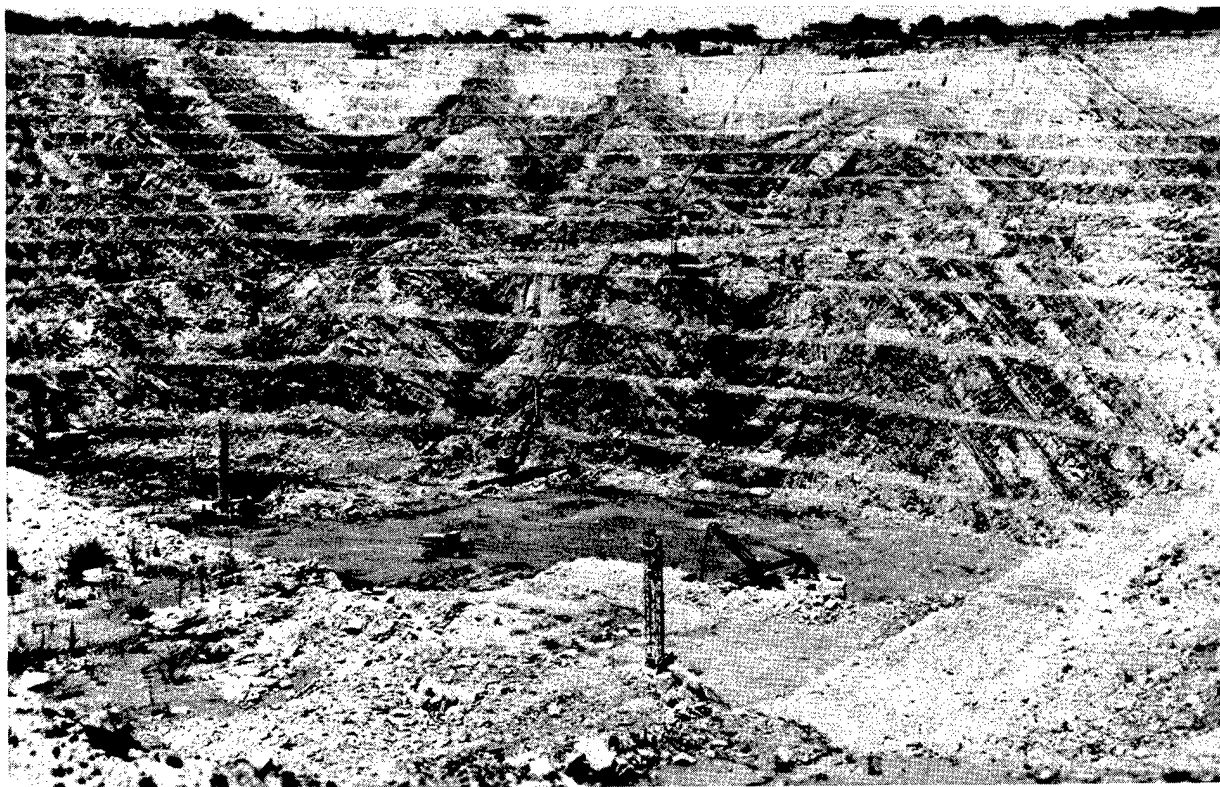


Figure 2.8 Chambishi Mine; a deep excavation in level terrain.
(Contributed by Roan Selection Trust, Ltd.)

2.8 INCLINATION OF EXCAVATED SLOPES

The average inclinations of excavated slopes listed in Table 2.1 range from 10 to 90°. The design of these slopes has been based largely upon engineering judgment, drawing upon experience in similar media and the use of "cut and try" procedures. Methods of defining slope stability parameters and applying them to problems of rock slope design are still in the formative stage.

The inclination of excavated slopes, and especially those of open pit mines, may also be governed by a number of factors such as configuration of the ore body, and requirements of the haulage systems, which are independent of slope stability considerations.

2.9 SLOPE STABILITY

Where available data indicate that a slope has been unstable, this fact has been noted in Table 2.1. In evaluating the significance of these cases, however, one must consider differences in failure criteria for various types of excavations. Evaluation of slope stability involves judgment based upon specific engineering requirements. Slope

adjustments which would be regarded as a failure in one situation might be acceptable under a different set of engineering requirements.

Rock bursts or rock falls which would constitute a serious engineering problem in a dimension stone quarry, for example, might be of no significant consequence in a large open pit mining operation. Rock slides which would seriously interfere with operations in one situation might be used to advantage in another as a means of collecting rock for removal. The surficial slide areas shown in Figure 2.9 might constitute an engineering problem, but they pose no immediate threat to the usefulness of this civil excavation.

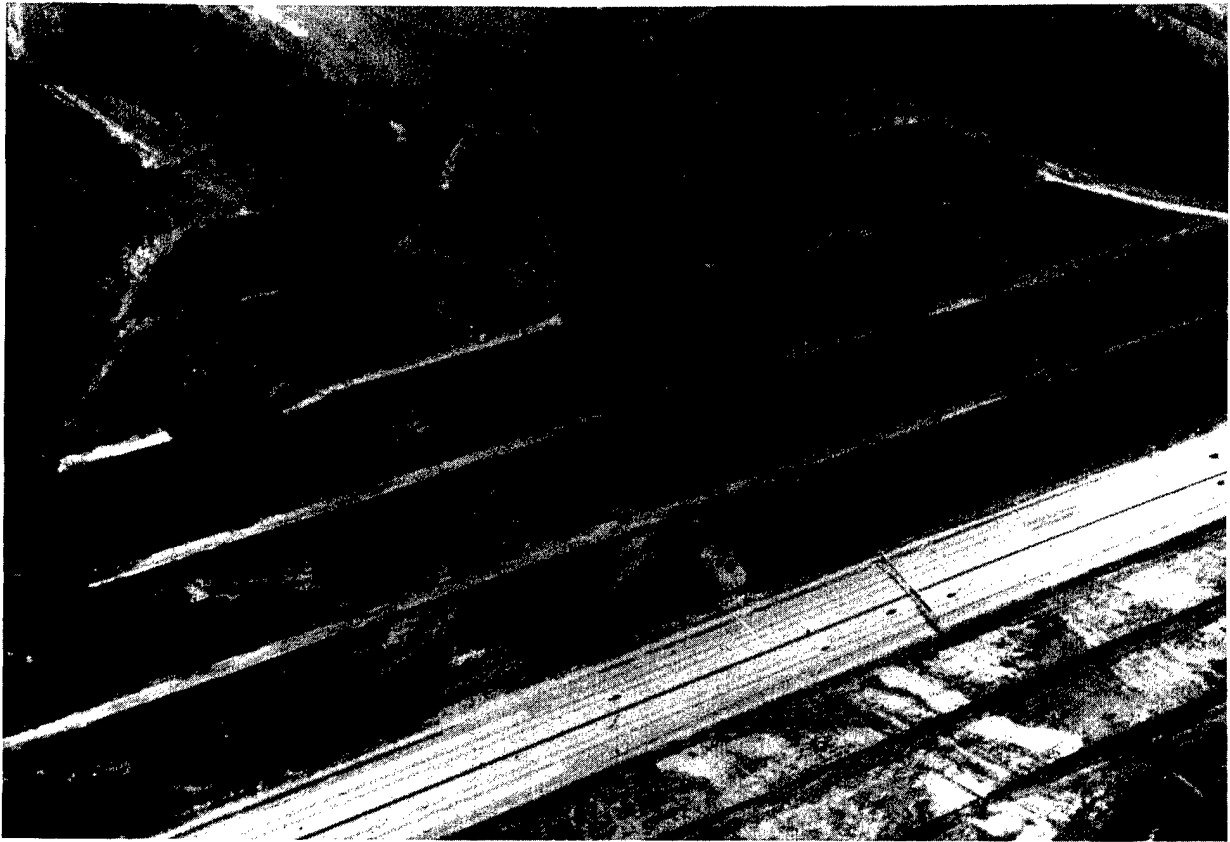


Figure 2.9 Carquinez Cut; oblique aerial view of southern wall, showing surficial slide.

Slope adjustments involving sloughing, rock bursting, or sliding of only one bench have not been noted as instabilities in Table 2.1. Of several cases of instability described in Appendix A, only two (at the East Jersey Pit* and the Mulholland Cut*) involved slope failures which destroyed the usefulness of the excavations for their intended purposes.

CHAPTER 3

ANALOGY TO NUCLEAR CRATER SLOPES

An understanding of the similarities and differences between conventional and nuclear excavations must be established before characteristics of conventionally excavated slopes can be used to assist in judging the stability of nuclear crater slopes. In Chapter 3, conventional and nuclear excavations are compared with respect to four major characteristics: mechanics of excavation, geology, physical environment of the excavation, and geometry.

3.1 MECHANICS OF EXCAVATION

The sharp contrast in mechanics of excavation is the most apparent and probably also the most significant dissimilarity between nuclear and conventional excavations. Consequently, even when features are found in conventional excavations which resemble characteristics of nuclear craters, the validity of any analogies may be limited by the extreme dissimilarity in modes of formation.

3.1.1 Degree of Control. Blasting operations in connection with conventional excavations are usually carried on in relatively small increments over a span of time totaling up to several years in civil projects, and up to 60 years or more in such open pits as the Bingham Canyon Mine,* and the Kiruna Pit.* The blasting program in such an operation, therefore, is a closely controlled process which can be modified at any time to meet changing requirements imposed, for example, by undesired slope movements, or by excess or insufficient breakage.

Nuclear excavation, in contrast, is accomplished essentially instantaneously. Experience to date indicates that the shape of the resulting excavation is a function of the cratering characteristics of the geological medium. The size of the excavation is a function of the explosive yield of the nuclear device, the depth of burst, and the properties of the geological medium. The degree of control which can be exerted over the configuration of a nuclear excavation, therefore, is limited.

3.1.2 Blast Fracturing and Bulking. The extent of blast fracturing and resultant bulking by explosive detonations is dependent upon (1) the lithology and structure of the geological medium, (2) the total explosive yield, and (3) the placement of explosive charges relative to one another and to the surface.

In smooth-wall blasting which is used extensively in civil projects, a line of closely spaced drill holes is placed along the designed outline of the excavation. Detonation of an array of light charges in these holes is just sufficient to promote

fracturing in the plane of the holes. Few fractures extend into the wall perpendicular to this plane (Figure 3.1, Part A). Subsequently, the material is blasted and excavated in lifts back to this presplit surface. The presence of the presplit surface inhibits propagation of fractures into the wall rock during excavation blasting. In this manner walls of the excavation are practically undamaged by subsequent blast fracturing, and steep inclinations can therefore be attained. The same technique is applied in some dimension-stone quarries where lines of drill holes are blasted with light charges to rough out the blocks.

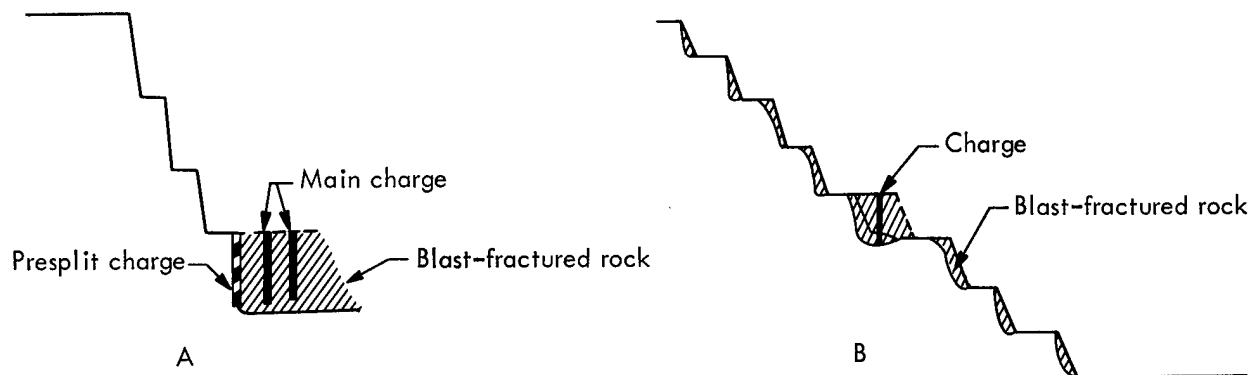


Figure 3.1 Schematic diagram illustrating charge placement and blast effects in (A) smooth blasting, and (B) conventional bench blasting.

In conventional open pit mining no presplitting is undertaken. Most of the useful work of the explosive in bench mining is performed in loosening the ore to permit mechanical removal; however, that portion of the energy of the charge that fractures the rock is not undesirable. This fracturing and reduction of rock size helps in reducing crushing costs at a later time. The miner is not greatly concerned about the direction in which fracturing extends. Blast hole patterns are designed to concentrate fracturing in the immediate mass being mined; however, blast-fractured rock remaining (Figure 3.1, Part B) after the loosened material has been removed poses no problem and is simply included in the subsequent round.

In explosive cratering little or no control can be exercised over the extent of the blast-fractured zone produced by a nuclear device or high explosive charge of given yield. Studies (References 274, 281) have been conducted on the shape and extent of fractured zones around both nuclear and high explosive craters in rock (Figure 3.2) resulting from charges with equivalent yields as great as 0.42 kt.

Craters produced by a row of five, 20-ton high explosive charges (Reference 274) (Figure 3.3) and by single 20-ton charges (References 252, 275) were found to be surrounded by a zone of rock with an increased effective porosity which approaches that of fallback. These studies have shown that blast fracturing and bulking in basalt extends laterally as much as one crater radius from the true crater boundary. Assuming a geometrical similarity, one can expect blast fracturing and bulking to extend for hundreds

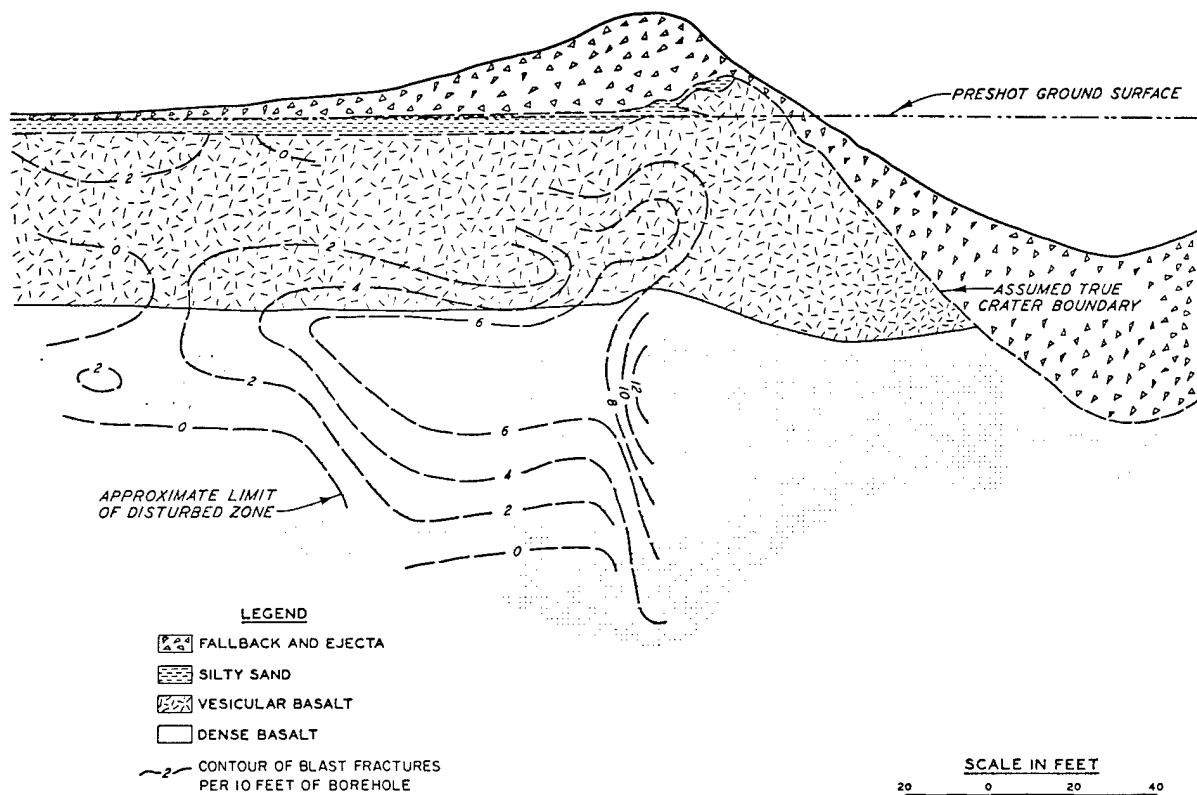


Figure 3.2 Dugout high explosive row crater in basalt; representative profile showing distribution of blast fractures. (Adapted from Reference 274, Lutton)

of feet from the true crater boundary in large nuclear craters. Particle size distribution of rock fragments produced by a nuclear cratering detonation and by conventional bench blasting are compared in Figure 3.4.

In summary, each blasting procedure will produce a distinct blast-fractured and bulked zone in the remaining wall rock. Smooth-wall blasting will result in a zone a few inches in thickness, and bench mining will result in a zone a few feet in thickness. Cratering will result in a zone extending up to hundreds of feet. Although little analogy is to be found here, extensive disrupted zones of other origin are found in a number of conventional excavations (Subsection 3.3.4).

3.1.3 Loading of the Rim. Much of the rock ejected by a nuclear blast comes to rest on the rim of the crater as a surcharge (Figure 1.3). An analogous condition, though not of comparable dynamic origin, exists adjacent to some open pits. The large amount of waste rock that either overlies or is intermixed with the ore must be removed to continue operation. This waste rock is usually dumped beyond the vicinity of the pit, but in some cases it may be necessary to dump waste on the rim area. Such cases bear a resemblance to the edge of a crater. In Figure 3.5 a thick layer of waste may be seen at one side of the Magcobar Pit. The underlying sandstone and shale dip away from the pit and resemble the upturned beds of a true crater lip, as revealed in postshot

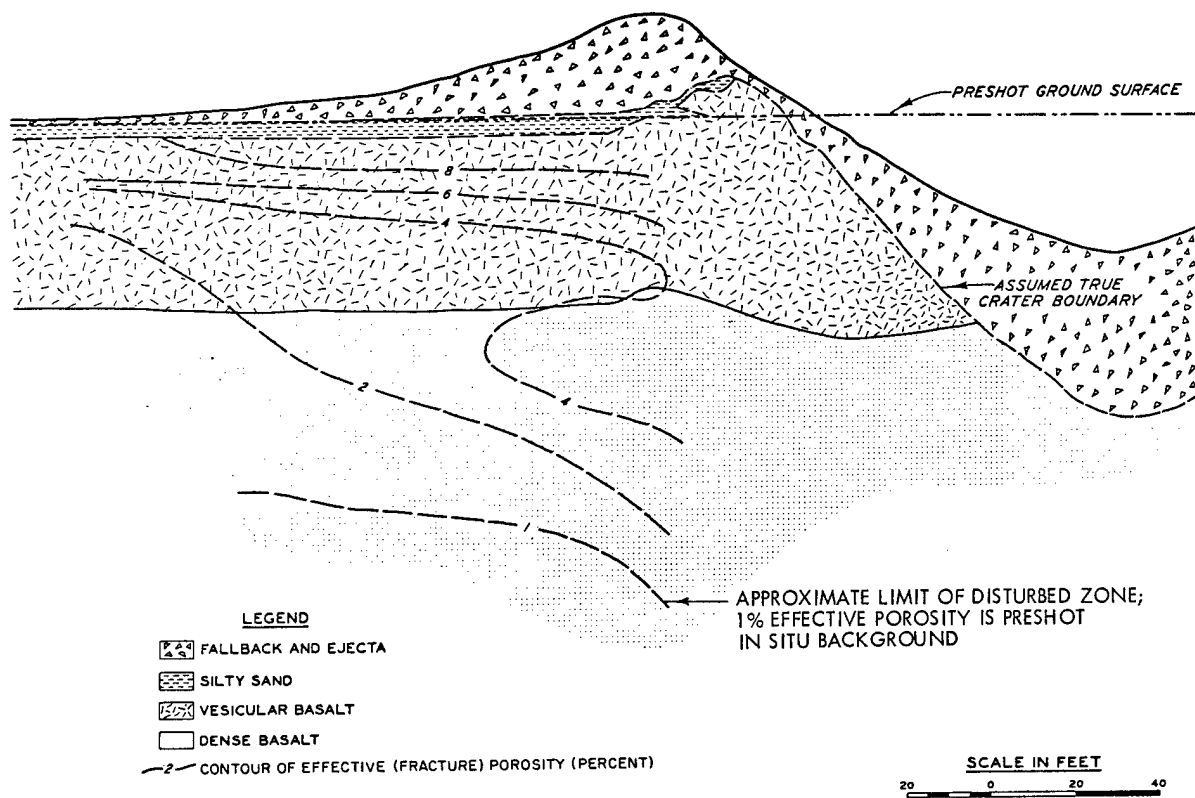


Figure 3.3 Dugout high explosive row crater in basalt; representative profile showing increased effective porosity due to blast fracturing and bulking.
(Adapted from Reference 274, Lutton)

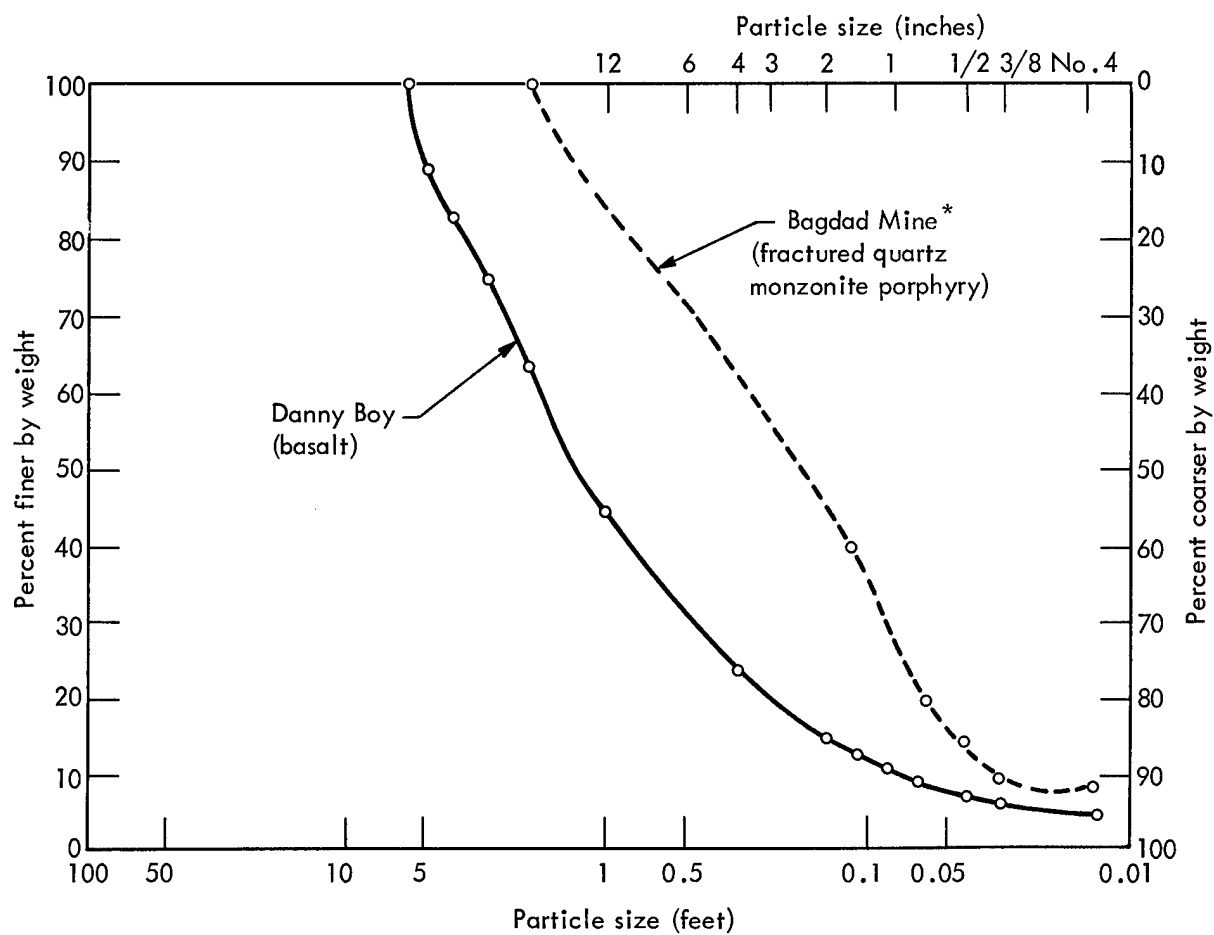


Figure 3.4 Particle size distribution curves for crater rubble (Danny Boy) and mine-run material produced by bench blasting (Bagdad Mine). (Danny Boy data adapted from Reference 281, Nugent and Banks, 1966. Bagdad Mine data contributed by Bagdad Copper Corporation.)

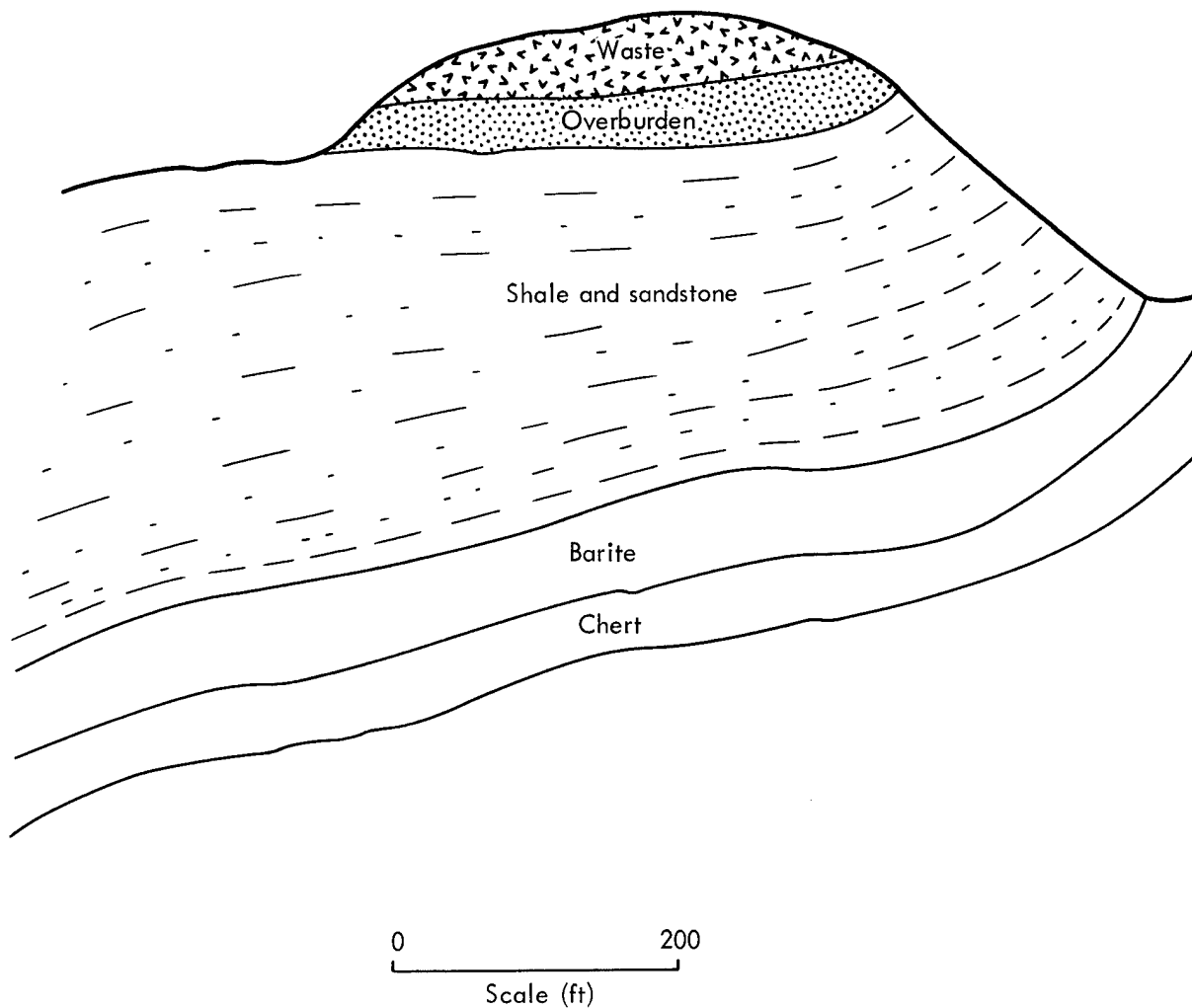


Figure 3.5 Magcobar Pit; representative section showing analogy to crater lip.
(Adapted from Reference 116, Scull, 1958)

investigations (References 252, 290) of craters in initially horizontal strata. In this and a few other localities such as the New Cornelia Mine,^{*} an analogy can be drawn, but in most cases there is no feature of conventional excavations corresponding to the ejecta lip of an explosion crater.

3.1.4 Adjustment of the Water Table. Any excavation which is carried below the depth of the water table creates a sump into which ground water will flow. Such drainage into an excavation may cause stability problems, particularly if the rate of excavation greatly exceeds the rate of drawdown.

Except for the fact that the cratering process rapidly alters the surrounding medium, the essentially instantaneous character of explosive excavation might be expected to maximize slope stability problems resulting from both a high water table and a rapid

drawdown. The available evidence, however, indicates that increased postshot permeability in the zone of bulking may offset these problems by permitting a rapid lowering of the water table, thereby reducing seepage forces.

Because of their slow rate of deepening relative to ground water drawdown, conventional excavations are poor analogs of proposed nuclear excavations, with respect to the influence of ground water upon slope stability.

3.1.5 Dynamic Loading. Dynamic stresses imposed upon adjacent media by a cratering detonation greatly exceed those imposed upon the slopes of conventional excavations during their incremental development. Conventional excavations, therefore, offer no analogy to nuclear excavations in this respect.

3.2 GEOLOGY

Although geological analogies can be found between conventional and nuclear excavation sites, they must be interpreted with caution. Analogies or resultant judgments based solely upon preshot geological conditions may have little or no bearing on the stability of a crater because of the disruption of preshot structure by the cratering process.

3.2.1 Reproducibility of Geological Setting. Individual geological features (e. g., rock type, folding or faulting, and orientation of structure relative to excavated surface) of existing excavated slopes can probably be compared in detail to corresponding preshot features of proposed nuclear excavation sites. The total preshot geological setting of an anticipated nuclear excavation could, in most cases, be matched with respect to its gross lithological and structural features, but it cannot be matched in all details. The precise combination of geological characteristics which influence the stability of any given slope is probably unique and not reproducible. The importance of such analogies with respect to slope stability is open to question, however, because of the destruction or radical alteration of preshot geological setting by a cratering event.

3.2.2 Geological Structure. As noted in Section 2.8 the majority of slope examples cited are mining excavations which are generally located in altered and structurally complex rocks. The rock alteration which has accompanied ore formation at most of these localities has, in many instances (e. g., the Berkeley Pit* and the Esperanza Mine*) diminished the strength of the original rock.

The dimensions of the largest proposed nuclear excavations may exceed those of most existing excavations by order of magnitude, although certain excavations such as the Bingham Canyon Mine,* the Bagdad Mine,* and the Penrhyn Quarry are comparable in size to the largest anticipated nuclear craters. Therefore, in comparing a conventional and a nuclear excavation in the same geological medium, it is likely that the total number of structural elements (e. g., faults, joints, folds, bedding planes, and flow layers) encountered will be somewhat greater in the case of the nuclear excavation, and the spacing of such structures relative to the overall dimensions of the excavated

slopes will be substantially less. Here again, however, the contrast between preshot and postshot characteristics must be considered.

3.2.3 Weathered Zone. A more or less decomposed weathered zone, varying from a few inches to several hundred feet in thickness, generally overlies sound bedrock. The nature and vertical extent of weathering are dependent upon both climate and bedrock lithology. In general, however, the depth of the weathered zone and the degree of decomposition are greatest in warm, humid climates.

Most of the major copper deposits of the western United States are overlain by thick zones of weathering which developed at a time when the climate of the region was probably more humid.

Because of the contrast in size between most conventionally excavated slopes and the slopes of proposed nuclear excavations, it is likely that the ratio of weathered-to-unweathered rock in a given locality will be substantially lower in the larger nuclear excavations. The mixing of weathered and unweathered materials in crater rubble may be of more significance than the relative volumes of such material.

3.2.4 Ground Water. The depth of the water table or zone of saturation below the ground surface ranges from zero to more than 1,000 feet. Ground water conditions found at proposed nuclear excavation project sites can be matched to those at conventional excavation sites.

As was discussed in Subsection 3.1.4, the effect of specific ground water conditions upon a nuclear excavation may not be the same as the effects of these same conditions upon a conventional excavation, because the postshot conditions adjacent to a nuclear crater are significantly different from preshot conditions.

3.3 PHYSICAL ENVIRONMENT

3.3.1 Seismic Activity. Seismic forces acting upon excavated rock slopes may include both natural and artificially induced impulses. In regions of comparable seismicity, the magnitude of earthquakes acting upon an excavated slope will be independent of the excavation technique. Some distinction may possibly be seen, however, between the response of conventional and nuclear excavated slopes to identical seismic loadings because of the contrasting material properties.

Seismic impulses generated by nearby blasting are experienced repeatedly by the slopes of conventional excavations. This situation, however, provides little analogy to the nuclear excavation case in which one crater would be subjected to fewer but significantly larger seismic loadings from the detonation of subsequent charges.

None of the slopes tabulated in this report has experienced stability problems attributed to either natural or artificially induced seismic loadings. In one instance, a 50-ton explosive charge was detonated behind an unstable slope at the Boron Mine* in an effort to accelerate movement, but no significant effects were obtained.

3.3.2 In Situ Stresses. In situ stresses surrounding a nuclear excavation may be completely unlike those surrounding a conventional excavation of comparable configuration at the same site because of the stress modification produced by blast fracturing, as was described in Subsection 3.1.2. Experience at the Fletcher Quarry* indicates that the adverse effects of high in situ stresses can be relieved by isolating the affected pit wall segment from its bedrock matrix. This stress relief process, which is accomplished by calyx drilling, may be achieved on a far greater and more effective scale in the blast-fractured zone surrounding a nuclear excavation.

3.3.3 Climate. Climate is directly or indirectly responsible for many of the geomorphic changes that take place in a slope. The weathering and erosion processes which produce these changes are time-dependent and, therefore, will have begun to act on an excavated slope during its period of existence. In some cases these processes proceed so slowly as to be insignificant. Elsewhere the effects of these processes may be controlled through the application of sound engineering design. For example, channels may be cut across portions of the slope that are in danger of carrying an excessive run-off. Slopes in cold climates may be flattened to avoid anticipated raveling due to intensive freeze/thaw action.

Some weathering and erosion processes may be accelerated by the greatly increased surface area exposed in the rubble zones of a nuclear excavation. On the other hand, the rubble slopes of a nuclear crater may approximate a long-term equilibrium condition in a conventional excavation, although the formational process is entirely dissimilar.

The effects of various long- and short-term weathering and erosion processes upon cohesionless material are being investigated as part of the continuing U. S. Army Corps of Engineers nuclear excavation research effort.

3.3.4 Rubble Zones. The physical properties (e. g., strength and particle size) of the thick zone of fallback within a nuclear crater (Figure 3.6) are grossly different from those of undisturbed bedrock. This rubble acts upon the underlying bedrock, both as a surcharge and as a buttress. No comparable mass of rubble is formed in conventional excavation processes; however, extensive rubble zones have been encountered in a number of conventional excavations. These include fault and subsidence breccias, and accumulations of sloughed material.

In some cases (e. g., Berkeley Pit*) engineering designs have been modified to provide lower slope angles in areas where such rubble zones are encountered. The available data, however, suggest that slopes in such rubble may be more stable than those in intact rock. At the Live Oak Pit,* stability problems are reported to be less frequently encountered in material that has been previously disrupted by underground block-caving. According to the operating company the block-caved rock has lost its original, unfavorable structure, and has become a relatively stable mass.

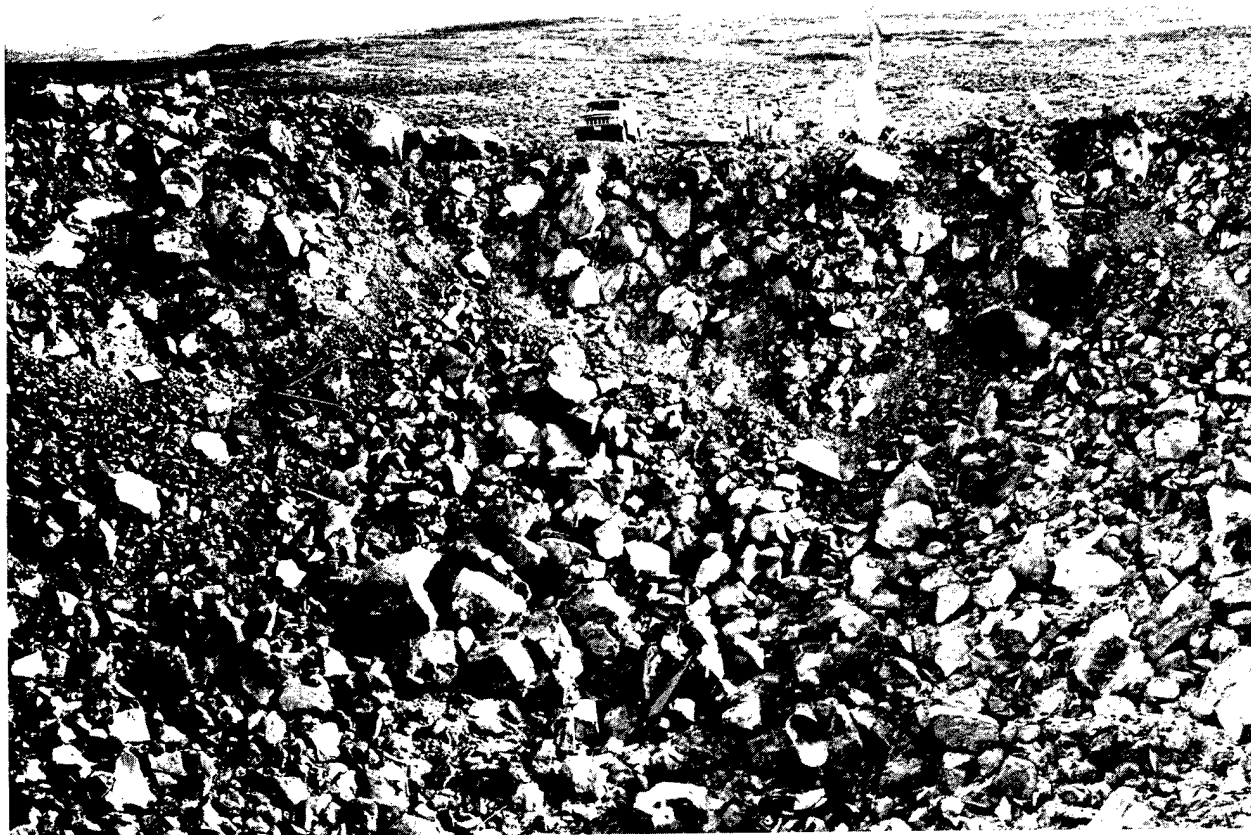


Figure 3.6 Fallback rubble in Danny Boy; 0.42-kt nuclear crater in basalt.

The use of rubble buttresses on intact rock slopes in the Mulholland Cut,^{*} provides an additional example of disrupted material comparable to fallback having a favorable effect upon stability.

3.4 GEOMETRY

As discussed below, similarity of shape is one of the best analogies between conventional and nuclear excavations.

3.4.1 Size of Excavation. The size of conventional excavations ranges up to about 1 mile in width or diameter and up to about 3 miles in length. Within this size range a number of analogs for nuclear excavations can be found.

For many years the Panama Canal was the largest man-made excavation in existence. A total of about 300 million cubic yards of material was removed up to the date of its completion in 1912. By 1939 this record had been surpassed by the continually expanding Hull-Rust-Mahoning Mine at Hibbing, Minnesota. By 1956 this mine had an area of 1545 acres, with a length of about 3-1/4 miles, a width from

1-1/2 to 1 mile, and a maximum depth of 520 feet (Reference 159). Between 1895 and the end of 1956, about 1 billion tons of ore and waste had been removed from the pit.

The Bingham Canyon Mine* (Figure 3.7) is about 7,000 feet in width and 12,000 feet in length, and has a maximum slope height of approximately 2,300 feet (comparable to the apparent crater depth plus apparent lip height of a 10-Mt crater in basalt, as shown in Figure 3.8). The apparent total depth of this pit is somewhat exaggerated,

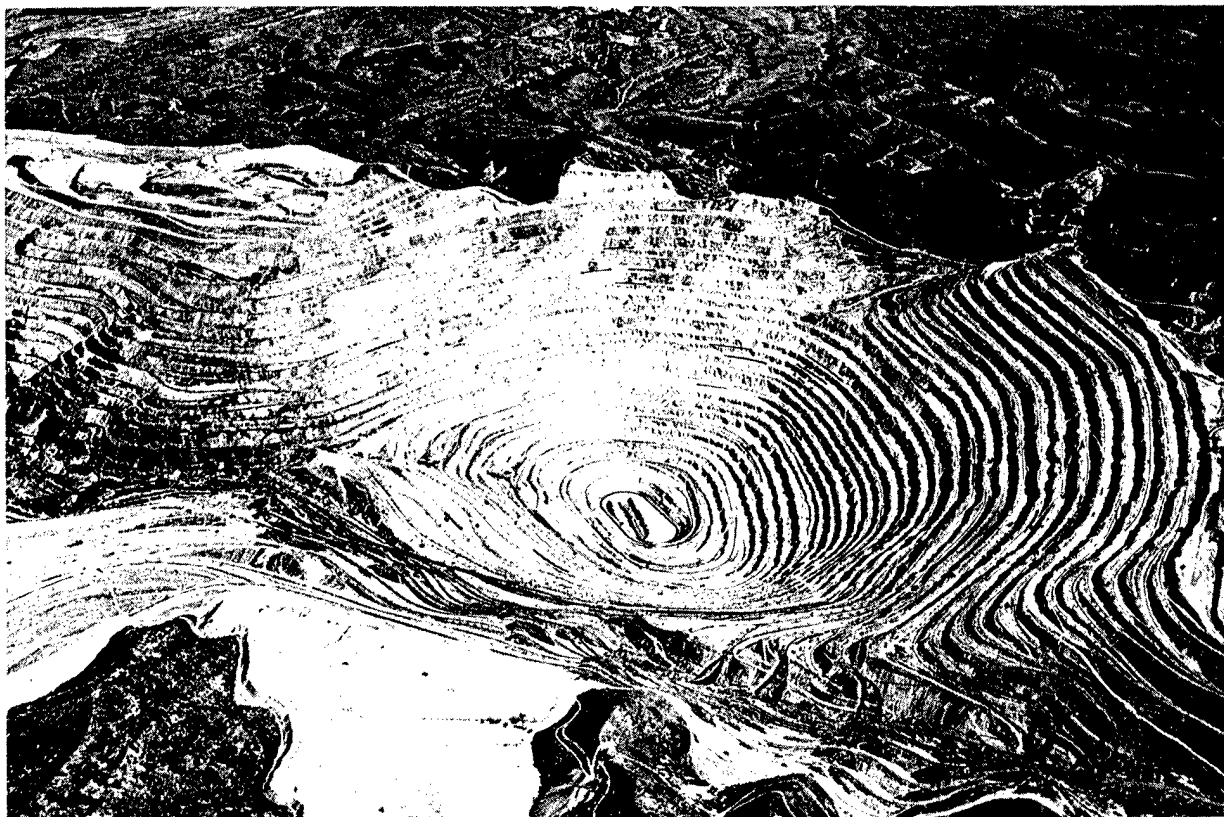


Figure 3.7 Bingham Canyon Mine; aerial view looking eastward; dimensions are comparable to those of a 10-Mt nuclear crater in basalt. (Reproduced by special permission of Bonneville News Company)

because the pit occupies the walls of a former canyon and much of its depth is due to natural erosion prior to mining. Nevertheless, depths of excavation, as defined in Section 2.7, reach a maximum of about 1,500 feet at this locality.

In addition to these "largest pits," there are many other mines, several of which are listed in Table A.1 and described in Appendix A, with sizes comparable to those of large nuclear craters.

3.4.2 Shape of Excavation in Plan. The detailed irregularities of ore bodies are generally smoothed over by the economics of low-cost, large-scale mining. Where a protrusion from the ore body cannot be worked into the mining system economically, it is left in the ground. Similarly, a sizeable mass of barren rock may be mined rather than being left as a protrusion into the pit. The result of this smoothing process is that pits tend to be approximately circular or ovoid in plan, approximating the shape of a

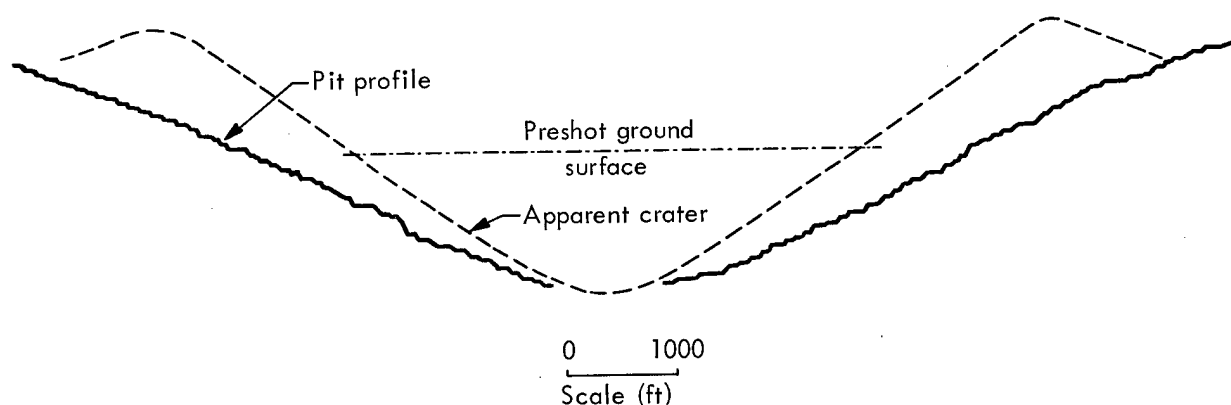


Figure 3.8 Representative profile of Bingham Canyon Mine, with superimposed profile of 10-Mt crater in basalt.

single-charge nuclear crater. Figure 3.9 illustrates the crater-like plan of the New Cornelia Mine.* Several open pit mines and civil excavations, however, approximate the linear configuration of a nuclear row-charge excavation. These include the Jackpile Mine (Figure 2.5), the Kiruna Mine,* and the Mulholland Cut.*

3.4.3 Slope Profile. As was discussed in Section 2.6, the overall profile of a slope may have an effect on its stability. Details of slope configuration, such as benches, may also be important.

The profile configuration of many conventionally excavated slopes is generally analogous to slope profiles of existing high explosive and nuclear craters (Figure 3.10). As was shown in Table 2.1, the average overall inclinations of excavated slopes in many open pit mines and civil excavations falls between 30 and 45°, bracketing the slope inclination of existing nuclear and high explosive craters.

Practically all civil and mining excavations have been benched for operational reasons and for safety. It has been found that bench mining is the most economical method for mechanical mining. Benching in civil projects facilitates drainage, inhibits erosion, interrupts the fall of loose blocks, and simplifies the mechanical excavation. No comparable feature exists in nuclear excavations.

3.4.4 Original Topography. Conventional excavation sites can be found which will match the gross pre-excavation topography at any proposed nuclear excavation site. Some examples are shown in Figures 2.7, 2.8 and 3.7. The details of topography, like those of geology, however, cannot be duplicated in an analogy, because the variety of possible combinations is too great.

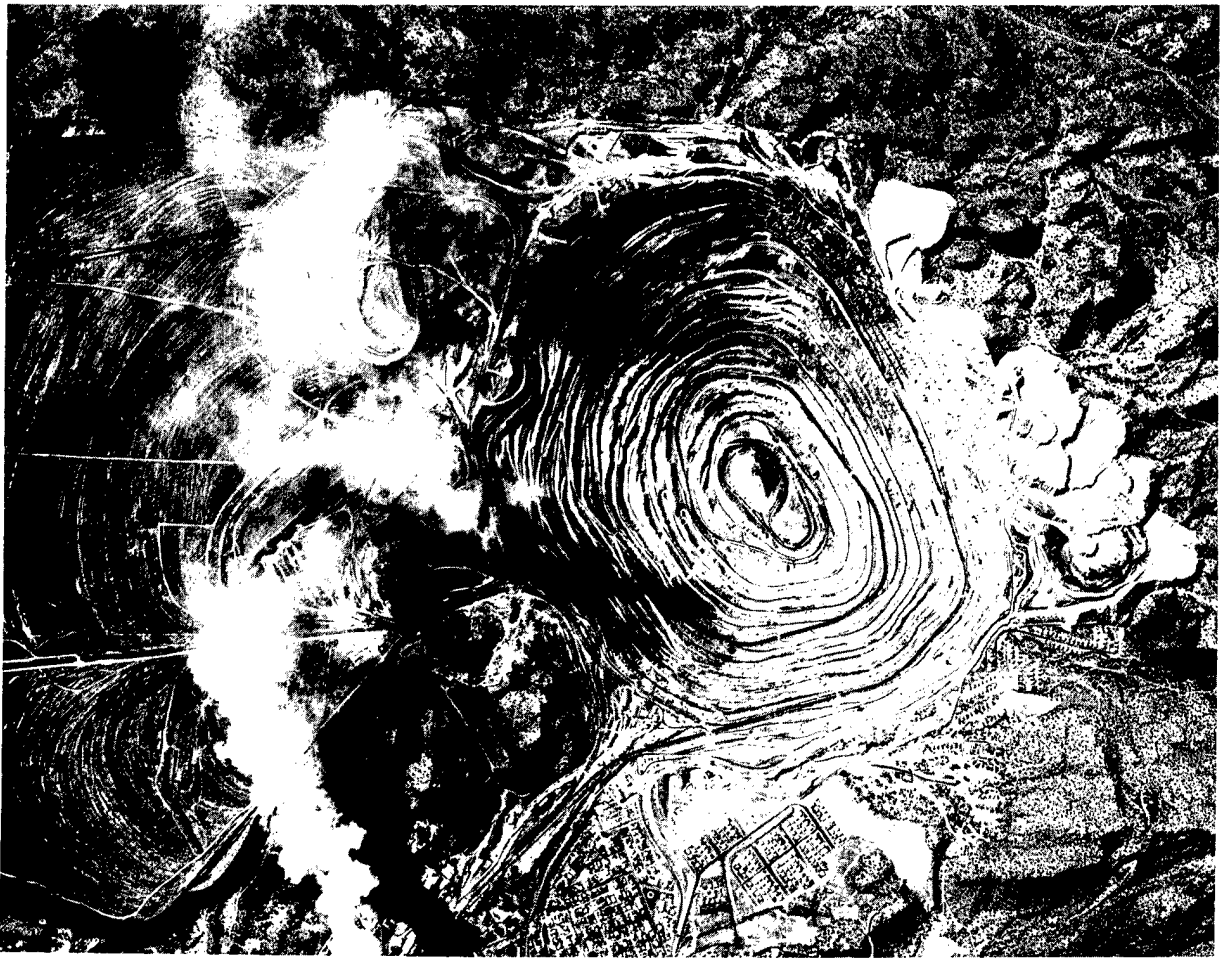


Figure 3.9 New Cornelia Mine; vertical aerial view showing craterlike plan.
(Contributed by Phelps Dodge Corporation)

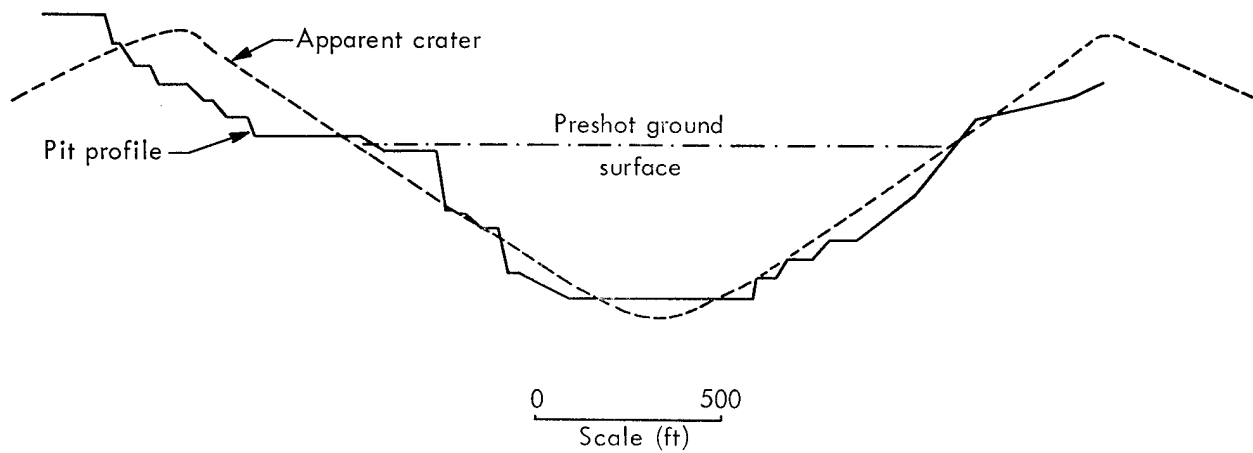


Figure 3.10 Representative profile of Bagdad Mine, with superimposed profile of 200-kt crater in basalt.

3.5 SUMMARY OF ANALOGIES

Possible analogies between conventional and nuclear excavations have been identified and evaluated in Chapter 3. In a few instances, particularly with regard to geometry, fairly close analogies are to be found. Qualitative analogies to rubble zones and rim loading are suggested, but the available data do not permit a detailed quantitative evaluation. In the case of blast-fracturing and bulking, dynamic loading, and effects of ground water, no general analogies can be identified. Table A.1 summarizes analogies for the excavations described in Appendix A.

CHAPTER 4

ANALYSIS OF RESULTS AND FUTURE RESEARCH RECOMMENDATIONS

This chapter indicates how selected quantitative data from this and future related studies can be used to improve understanding of the long-term stability of nuclear craters as engineering structures. The results of a statistical analysis of slope data are given. Specific areas are identified for further study in order to assist current research in engineering properties of nuclear craters.

4.1 RESULTS OF A STATISTICAL ANALYSIS OF SLOPE DATA

Of the various factors considered in this report, three that appear to influence slope inclination and stability are amenable to a simple statistical analysis. These are structural pattern, mass strength, and slope height.

Table 4.1 indicates that a total of 285 slopes, those with a disordered structure have generally been cut to a greater overall inclination than those having a partially ordered or well-ordered structure.

TABLE 4.1 STATISTICAL ANALYSES OF SLOPE ANGLES IN TERMS OF VARIOUS PARAMETERS

		All Slopes					Stable Slopes					Unstable Slopes				
Parameter		No. of Items	Arithmetic Mean	Median	Standard Deviation	Mode	No. of Items	Arithmetic Mean	Median	Standard Deviation	Mode	No. of Items	Arithmetic Mean	Median	Standard Deviation	Mode
		degrees	degrees	degrees	degrees		degrees	degrees	degrees	degrees		degrees	degrees	degrees	degrees	
Structural Pattern	Partially ordered	131	40.5	37	15.6	42	111	40.7	37	16.0	42	20	35.6	32	12.0	44
	Ordered	113	43.4	39	16.2	44	73	44.5	40	17.2	43	40	41.6	36	14.2	32
	Disordered	41	49.1	45	14.4	44	35	49.0	43	15.4	42	6	50.1	--	--	--
Mass Strength	Hard	229	43.2	40	15.3	43	190	43.2	40	15.7	44	39	42.1	38	12.4	35
	Soft	56	40.6	35	18.9	31	29	42.9	35	16.8	28	27	38.3	34	15.0	33
Slope Height (feet)	100-299	115	47.4	42	18.1	43	83	49.0	44	18.8	44	32	43.6	41	15.9	44
	300-499	100	42.7	40	12.9	43	84	43.6	42	13.4	45	16	38.1	36	8.46	39
	500-699	32	40.1	38	11.1	42	26	40.5	39	12.0	42	6	38.1	37	6.0	36
	700-899	9	37.2	35	9.5	35	6	33.7	30	8.9	29	3	44.3	--	--	--
	≥ 900	18	26.5	24	9.0	24	15	27.0	24	9.0	24	3	24.0	--	--	--

NOTE: 1. Angles tabulated for unstable slopes may represent either inclinations prior to or following adjustment (differences in overall inclination before and after adjustment are generally insignificant).

2. This analysis is based on all slopes in Table 2.1 except
a. Jeffrey Mine (added to Table 2.1 after completion of statistical computations)
b. Postadjustment slopes where inclination is significantly less than original slope angle.

In considering the mass strength of 285 slopes, those in hard rock were found to have been generally cut to a steeper inclination than those in soft rock. Stable slopes in both hard and soft rock were found to have been cut at steeper average inclinations than unstable slopes.

A total of 274 slopes considered in Table 4.1 were divided into height classes of 100-299, 300-499, 500-699, 700-899, and ≥ 900 feet, for comparison with slope inclination. Stable excavated slopes, in general, are found to have been constructed with overall inclinations which vary inversely with slope height. This relationship is less apparent in the case of unstable slopes.

Experience to date indicates that craters in basalt are formed with slopes of approximately 35° . This inclination is equalled or exceeded by many stable excavated slopes considered in this report, particularly those less than 900 feet in height. Still, the fact that many of the slopes cited in Table 2.1 have proven to be unstable at angles of less than 35° , indicates that average values have little relevance in a consideration of specific cases. The stability of a specific slope will be dependent upon factors such as structural orientation and ground water conditions, as well as those considered in Table 4.1

4.2 SUBJECTS FOR FURTHER INVESTIGATION

Other possible analogs which may be related to slope stability should also be analyzed in a more quantitative manner, but adequate detailed data are not presently available. If quantitative data concerning the effects of weathering and construction operations were available, numerical strength parameters could be derived for specific excavated slopes. These could then be employed as a basis for formulating engineering judgments where full-scale nuclear cratering data are not available.

This study provides a basis for additional investigations of conventionally excavated rock slopes to aid in evaluating the stability of nuclear craters. Specific subjects worthy of further study include the effect upon slope stability of the following:

- a. Plan and profile configuration
- b. Structural disruption
- c. Weathering and erosion
- d. Waste deposition at the rim of an excavation
- e. Rapid dewatering.

Future investigations should be concentrated on slopes comparable in height to those of large nuclear craters (i. e., over 500 feet).

4.3 FUTURE RESEARCH

The continuing research program of the Corps of Engineers to study the engineering properties of nuclear craters provides for investigation of conventionally excavated slopes. The goal of this program is to understand the performance of nuclear craters as engineering structures over the useful lifetime of the project. It is believed that the studies outlined in Section 4.2 will assist in attaining this goal.

CHAPTER 5

CONCLUSIONS

On the basis of data presented in this report, the following conclusions are made regarding conventional-nuclear analogies.

- a. Definite analogies exist between conventional rock excavations and proposed nuclear excavations, particularly with regard to such surficial geometrical characteristics as shape, depth of excavation, slope height, and slope angle.
- b. Additional analogies involving lip loading and structural disruption are qualitatively suggested, but they cannot be quantitatively substantiated on the basis of the available data.
- c. The range of geological and environmental features represented by the localities cited in this report is sufficiently great to ensure that at least a general and qualitative analogy can be found for any specific geological or environmental feature of a proposed nuclear excavation site.
- d. Because of the limited quantitative data available, a substantial measure of engineering judgment is required to interpret and apply the analogies between conventional excavations and nuclear craters.
- e. The data presented in this report and the analogies established or suggested provide points of departure for future studies relating engineering experience with conventionally excavated slopes to the evaluation of stability in proposed nuclear excavations.

On the basis of a preliminary analysis, the following tentative conclusions are made regarding the stability of conventionally excavated rock slopes:

- a. Slope inclination appears to be a function of slope height, mass strength, and structural pattern.
- b. Geological structures which dip toward an excavation are a major cause of instability, particularly where ground water problems are also present.
- c. Structural disruption, as produced in block caving, may increase stability.

APPENDIX A
BRIEF DESCRIPTIONS OF SELECTED LOCALITIES

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TABLE A.1 CONVENTIONAL EXCAVATIONS DESCRIBED IN APPENDIX A AND THEIR POSSIBLE ANALOGY TO NUCLEAR EXCAVATION

Locality	Maximum Slope Height (feet)	Nuclear Yield to Produce Apparent Crater of Similar Depth (depth plus lip height)	Elongate or Linear Plan	Equidimen- sional Plan	Lip Loading	Rubble Zones	Cut Through Hill Mass
Bagdad Pit	800	200-500 kt	No	Yes	Yes	Yes	No
Berkeley Pit	600	100-200 kt	No	Yes	No	Yes	No
Bingham Canyon Mine	2,280	10 Mt	No	Yes	Yes	No	No
Boron Pit	400	100 kt	No	Yes	No	No	No
Castle Dome Mine	620	100-200 kt	No	Yes	No	No	No
Chambishi Mine	400	100 kt	No	Yes	No	No	No
Day-Loma and Frazier-Lamac Pits	318	<100 kt	No	Yes	No	No	No
East Jersey and Jersey Pits	480	100 kt	Yes	No	No	No	No
Esperanza Mine	420	100 kt	No	Yes	No	No	No
Fletcher Quarry	215	<100 kt	No	Yes	No	No	No
Kimbley Pit	500	100 kt	No	Yes	No	No	No
Kiruna Pit	750	200 kt	Yes	No	Yes	Yes	Yes
Live Oak and Thornton Pits	750	200 kt	No	Yes	Yes	Yes	No
Missabe Mountain Mine	425	100 kt	No	Yes	No	No	No
Mulholland Cut	370	<100 kt	Yes	No	No	Yes	Yes
New Cornelia Mine	730	200 kt	No	Yes	Yes	No	No
Pearl Handle and West Pits	500	100 kt	No	Yes	No	Yes	No
Sanford Lake Mine	800	200-500 kt	Yes	No	No	No	No
Steep Rock Lake District	550	100 kt	No	Yes	No	No	No
Toquepala Mine	1,100	1 Mt	No	Yes	No	No	No
United Verde Pit	1,000	500 kt-1 Mt	No	Yes	No	Yes	No

BAGDAD MINE
BAGDAD, ARIZONA (2)*

This open pit copper mine is located a short distance north of Bagdad, in west-central Arizona. The Bagdad Mine is the only major mining operation in the area, although there are a number of small base and precious metal mines and prospects in the vicinity which have been worked sporadically in recent years, and which may be in operation at the present time.

Topographic features of the Bagdad area include mountains, rolling hills, and lava-capped mesas dissected by deep canyons. The Bagdad Mine itself is situated in Copper Creek Canyon which lies between Sanders Mesa to the east and Copper Creek Mesa to the northwest. Natural relief in the local area is about 1,200 feet.

The climate at Bagdad is semiarid, with annual precipitation ranging from 7 to 26 inches and averaging about 14 inches. Maximum rainfall occurs from July through September, and again from December through March. Summer rains, which are commonly of the thunderstorm or cloudburst variety, may cause brief but torrential surges of stream flow. Winter precipitation, which is more evenly distributed, results in high but uniform rates of stream flow. The permanent water table lies about 100 feet below the original ground surface at the mine. Local temperatures range from a summer maximum of about 100°F to a winter minimum of about 10°F.

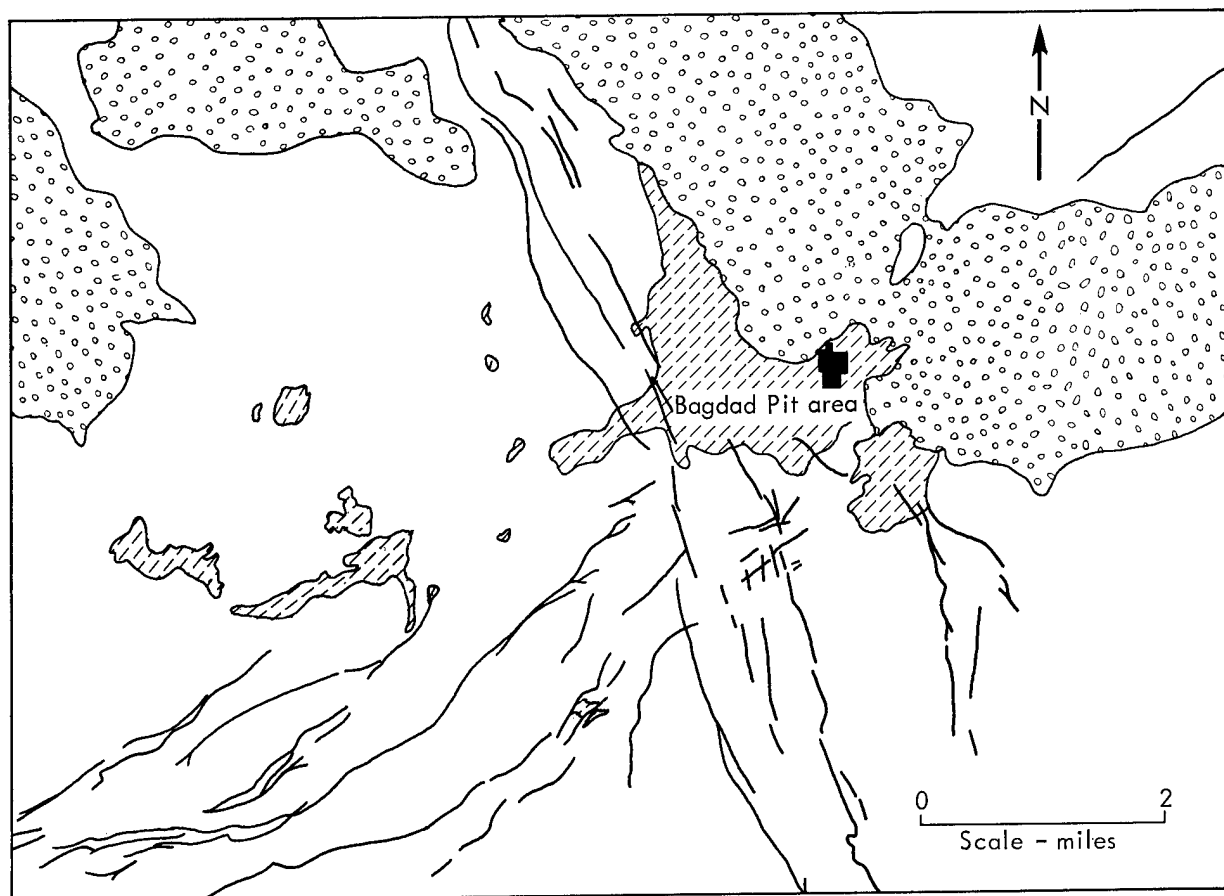
The bedrock in the vicinity of the Bagdad Mine consists of Precambrian metasedimentary, metavolcanic, and associated intrusive rocks of diverse composition. This "basement complex" has been covered by rhyolitic tuff, alluvial gravels, and basaltic flows, and has been intruded by rhyolitic dikes and quartz monzonite stocks. The major structure of the basement complex is that of an overturned synclinal fold which has been modified by high-angle faults. Some of these faults have served as channelways for later igneous intrusions. All rocks of the region are broken by faulting, though no displacements of major magnitude have been reported.

The Bagdad Mine is located at the intersection of two near-vertical shear zones, (cf. Figure A.1). Numerous diversely oriented fractures (Figure A.2) within the quartz monzonite stock have served as loci for primary sulfide mineralization and hydrothermal alteration (silicification and/or sericitization).

Although copper mineralization was discovered at Bagdad in 1882, systematic exploratory drilling was not undertaken until 1906. Underground mining of relatively rich massive sulfide ores was begun in 1929.

As was the case in most of the major "porphyry copper" deposits, the extensive low-grade ores of the Bagdad district were discovered at an early date, but their

*Numbers in parentheses refer to localities shown in Figures 2.1 and 2.2.



Legend

- | | | |
|-------------------------------|--------------------|---------------------------------|
| Conglomerate and basalt flows | Quartz monzonite | Outline of underground workings |
| Dikes | Pre-Cambrian rocks | |

Figure A.1 Generalized geological map; vicinity of Bagdad Pit.
(Adapted from Reference 3, Anderson, Scholz, and Strobell, 1955)

development was not undertaken until necessitated by diminishing reserves of higher grade ores. Thus, the operations of the Bagdad Mine evolved, as mining technology improved and ore grades diminished, from conventional stoping to block-caving during the late and early 1940's. Surface mining was begun in 1945, with ore being dumped into a glory hole for removal through underground haulageways. Finally, in 1947, underground operations were halted and the present large-scale, open-pit operation was undertaken.

The major ore body being mined in present open pit operations is a zone of supergene enrichment within an irregularly shaped stock of porphyritic quartz monzonite which intrudes granitic and metamorphic rocks of the basement complex. The zone of supergene enrichment averages 125 feet in thickness, and is inclined to the northeast at 10 to 15°. Weathering which produced the supergene ores has also given rise to a leached and oxidized capping up to 350 feet in thickness.

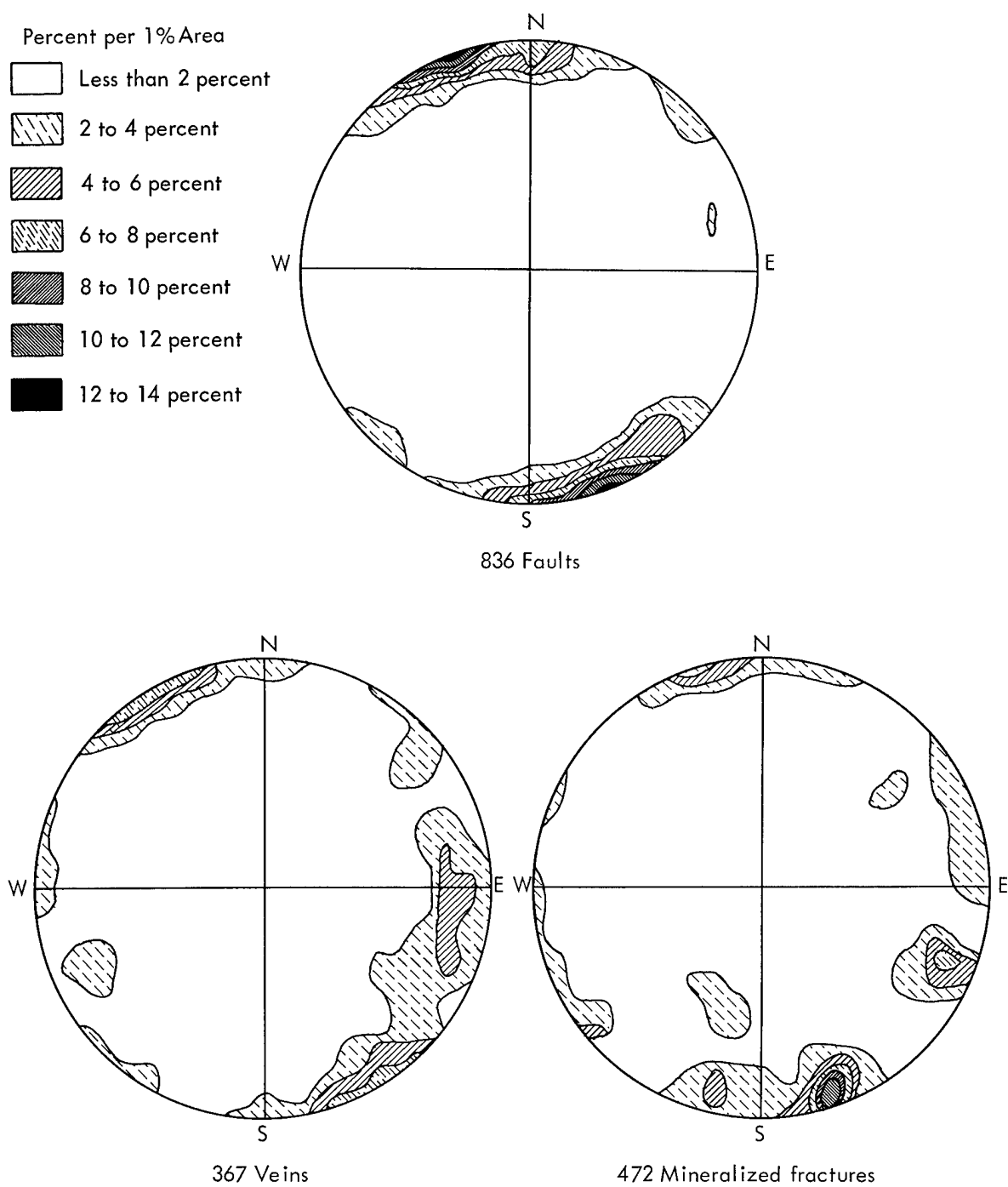


Figure A. 2 Bagdad Mine; stereograms (lower hemisphere projections) showing fault, vein, and fracture orientations. (Adapted from Reference 3, Anderson, Scholz, and Strobell, 1955)

The Bagdad Pit is roughly oval in outline, measuring approximately 3,000 feet along the major NE-SW axis and 2,500 along the minor NW-SE axis (Figure A.3). The pit has been excavated to a depth of about 900 feet. Pit walls are broken by benches ranging from 40 to 100 feet in width and spaced at uniform vertical intervals of 50 feet. Overall pit wall angles range from about 12 to 45° (Figure A.4). One stable pit wall segment in the western portion of the mine has a height of 900 feet and an average inclination of 45°.

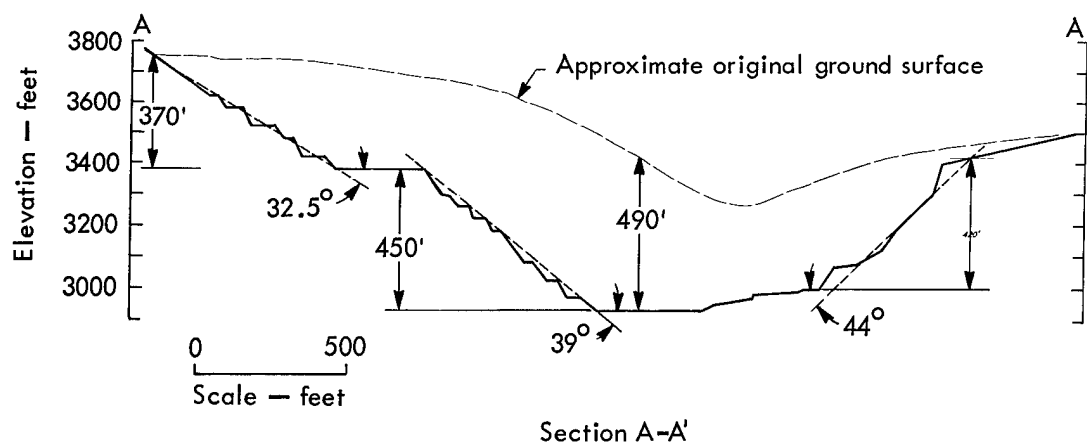
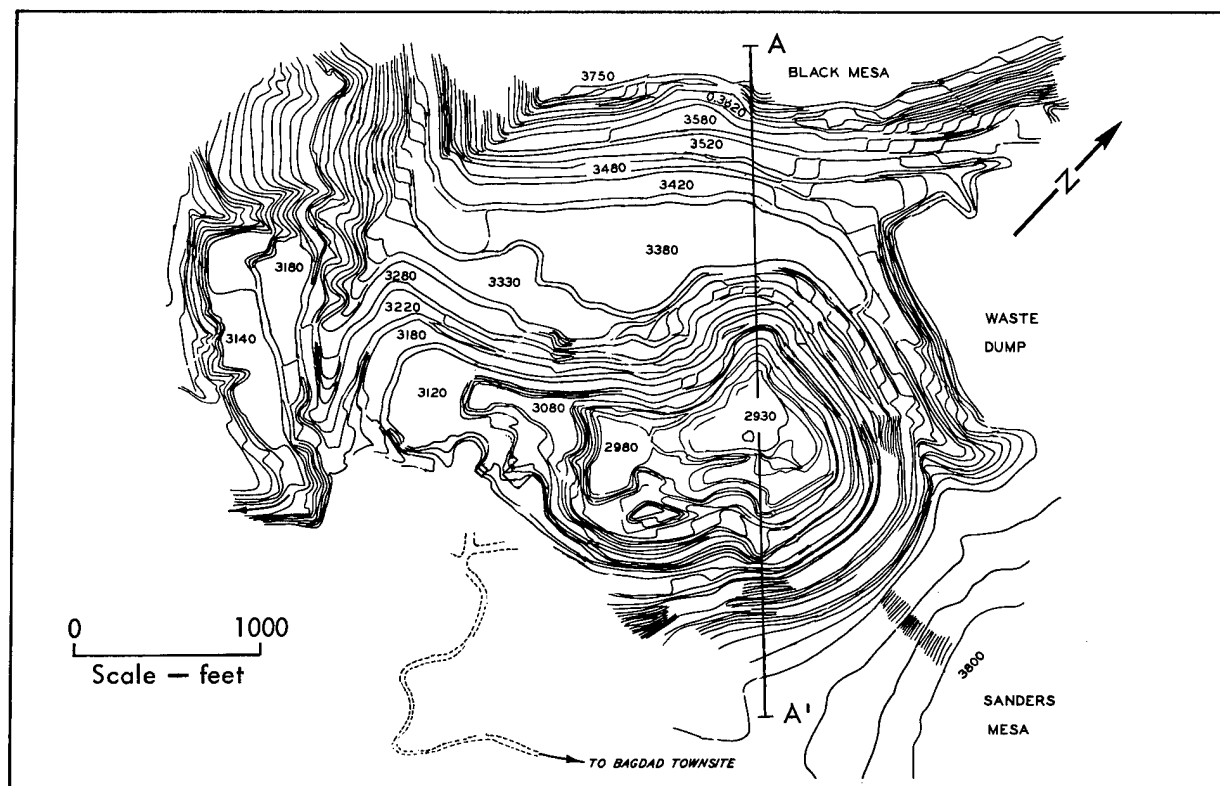


Figure A.3 Bagdad Pit; plan and representative profile. (Adapted from Reference 51, Hardwick and Jones, 1959)

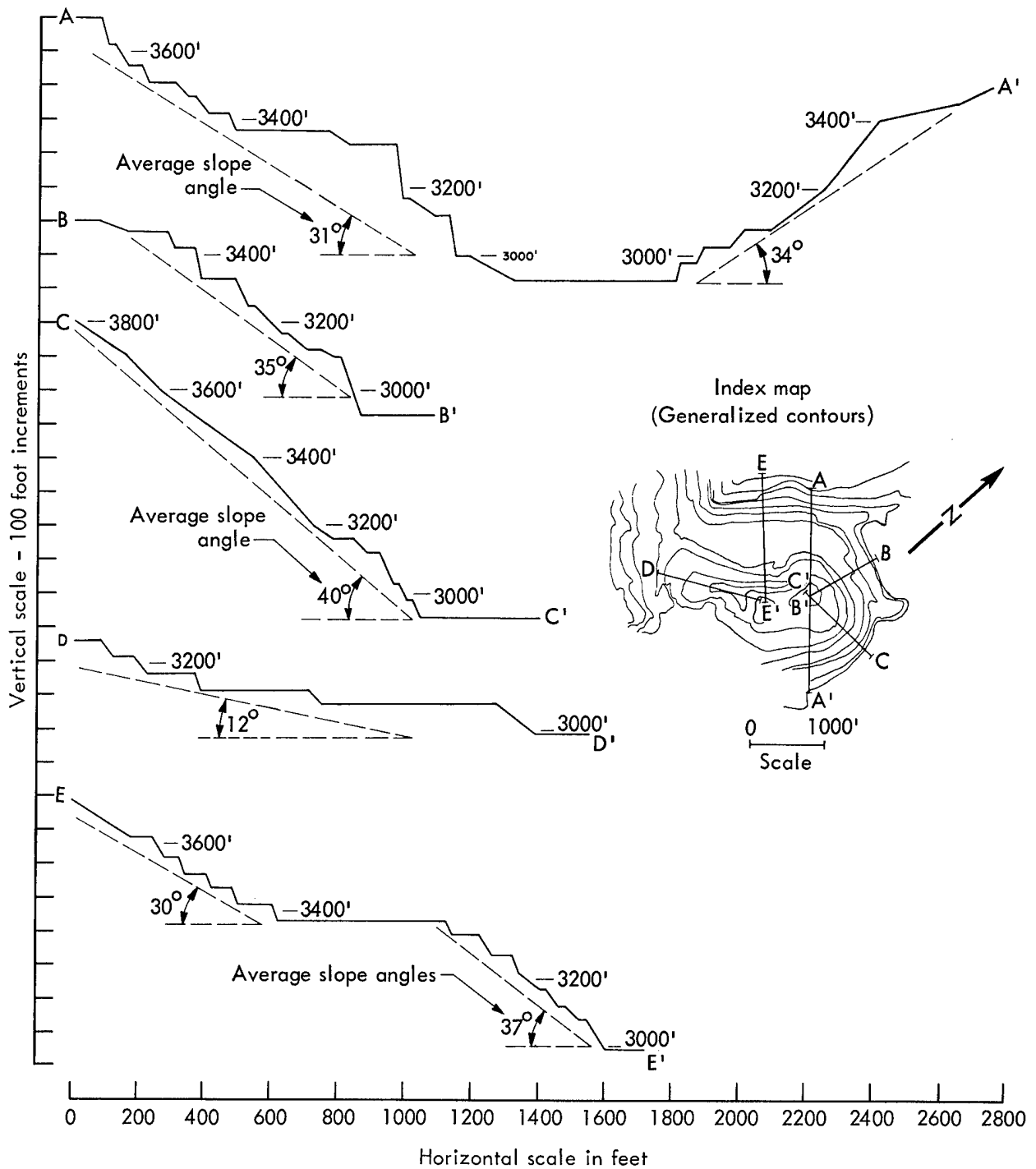


Figure A.4 Bagdad Pit; generalized plan and representative profiles. (Adapted from Reference 51, Hardwick and Jones, 1959)

Except for localized surficial deterioration of individual benches, no slope stability problems are reported at Bagdad, although several zones of block-caving subsidence are encountered within the pit excavation. The operating company reports that no special modifications of pit wall inclination are required in block-caved rubble zones. Special

precautions are taken to drain the pit rim area to avoid ponding of surface water near the pit.

References 3, 8, 47, 51, 151, 165 (cf. also Figure 3.10).

BERKELEY PIT BUTTE, MONTANA (4)

The Berkeley Pit is situated on the southeastern flank of Butte Hill, between the city of Butte and the unincorporated town of Meaderville. Although the Berkeley Pit is the only major open pit mine of the Butte district, a number of underground operations (involving both selective stoping of high-grade veins and block-caving of lower-grade ores) are still in progress in the immediate vicinity.

The Butte district is situated at an elevation of about 6,000 feet in a westward sloping area just west of the Continental Divide in the Rocky Mountains of western Montana. Higher peaks of the surrounding areas, which rise to elevations in excess of 8,000 feet, bear a coniferous forest cover. Lower slopes, such as those on which the city of Butte is situated, are open and grassy (though noxious smelter fumes have wiped out vegetation in much of the immediate vicinity of Butte). Valley floors, though occupied by permanent streams, are quite arid and barren.

The climate of the district is semiarid, with an average annual precipitation of 12 to 15 inches, occurring chiefly as snow during the winter and spring. Temperatures range from -30 to +90°F with an average annual temperature of about 50°F.

Little information is available concerning ground water levels in the Butte district. A few small springs are reported on the slopes of Butte Hill; but it seems probable that the general level of the water table has been lowered greatly over the years through drainage channels provided by the extensive network of underground mine workings.

Since 1864 when mining began in this district, the Butte ore deposits have supported one of the most active and prosperous mining districts in the world. Butte's "Richest Hill on Earth" has been honeycombed to depths of nearly 1 mile by more than 8,000 miles of underground workings, and has produced nearly 3 billion dollars worth of silver, zinc, lead, gold, and manganese.

Until recent years, mining at Butte was confined to relatively rich fissure veins which were selectively stoped throughout their length and depth. The presence of widespread low-grade mineralization was noted at an early date, but mining technology had not yet advanced to the point where these low-grade deposits could be profitably developed. By 1947, however, declining reserves of high-grade ore prompted initiation

of the "Greater Butte Project" to consider the feasibility of developing these low-grade deposits through the use of high-volume, low-cost mining methods.

Subsequently, major block-caving operations were begun within existing underground mines, and an intensive drilling program was undertaken to block out a possible open pit ore body. Finally, in 1955, excavation of a test pit was begun in the vicinity of the old Berkeley Shaft, and the results proved sufficiently favorable to warrant its expansion into a major producing mine.

The Butte district is situated just inside the margin of the Boulder batholith, a plutonic mass of intermediate composition which has intruded older andesites. The bedrock at Butte is a quartz monzonite which has been intruded by porphyritic, aplitic, and rhyolitic dikes. The bedrock is transected by a complex system of shear faults. These fissures, which have served as loci for hydrothermal alteration and ore deposition, can be grouped into two broad categories as follows: (1) a system of NE-SW trending faults which dip steeply to the southeast, and (2) a system of NW-SE trending faults which dip steeply to the southwest. A series of E-W trending tensional fissures, though numerically subordinate to the major shear systems, have been particularly important as loci of high-grade ore mineralization. Toward the eastern end of the district many major faults give way to systems of subsidiary fractures, and intersections of various fracture systems give rise to an exceedingly complex zone of anastomosing mineralized veinlets (Figure A.5).

In general, the various forms of hydrothermal wall rock alteration (in particular, sericitization and kaolinization) which have accompanied mineralization at Butte have reduced the density, the hardness, and the strength of the monzonitic pluton. Oxidation, leaching, and other weathering processes have further reduced the structural competence of the bedrock.

The Berkeley Pit is located within the complex zone of anastomosing fissures near the eastern end of the Butte district. There are, however, also a number of major through-going shear faults of both the NE-SW and NW-SE systems which cut across the pit area. Weathering of the mineralized monzonite has produced a barren leached capping 100 to 300 feet in thickness, and an underlying zone of supergene enrichment which ranges from 100 to 600 feet in thickness (Figure A.6).

The pit is approximately triangular in outline, with sides ranging from about 2,000 to 3,000 feet in length (Figure A.7). The southern and eastern walls of the pit have been cut to relatively high angles (32 to 46°, respectively), while the northwestern wall is more gently inclined (21°). Pit walls are broken by benches of variable width which are spaced at average vertical intervals of about 50 feet. Individual bench slopes range from about 25 to 75° (Figure A.8).

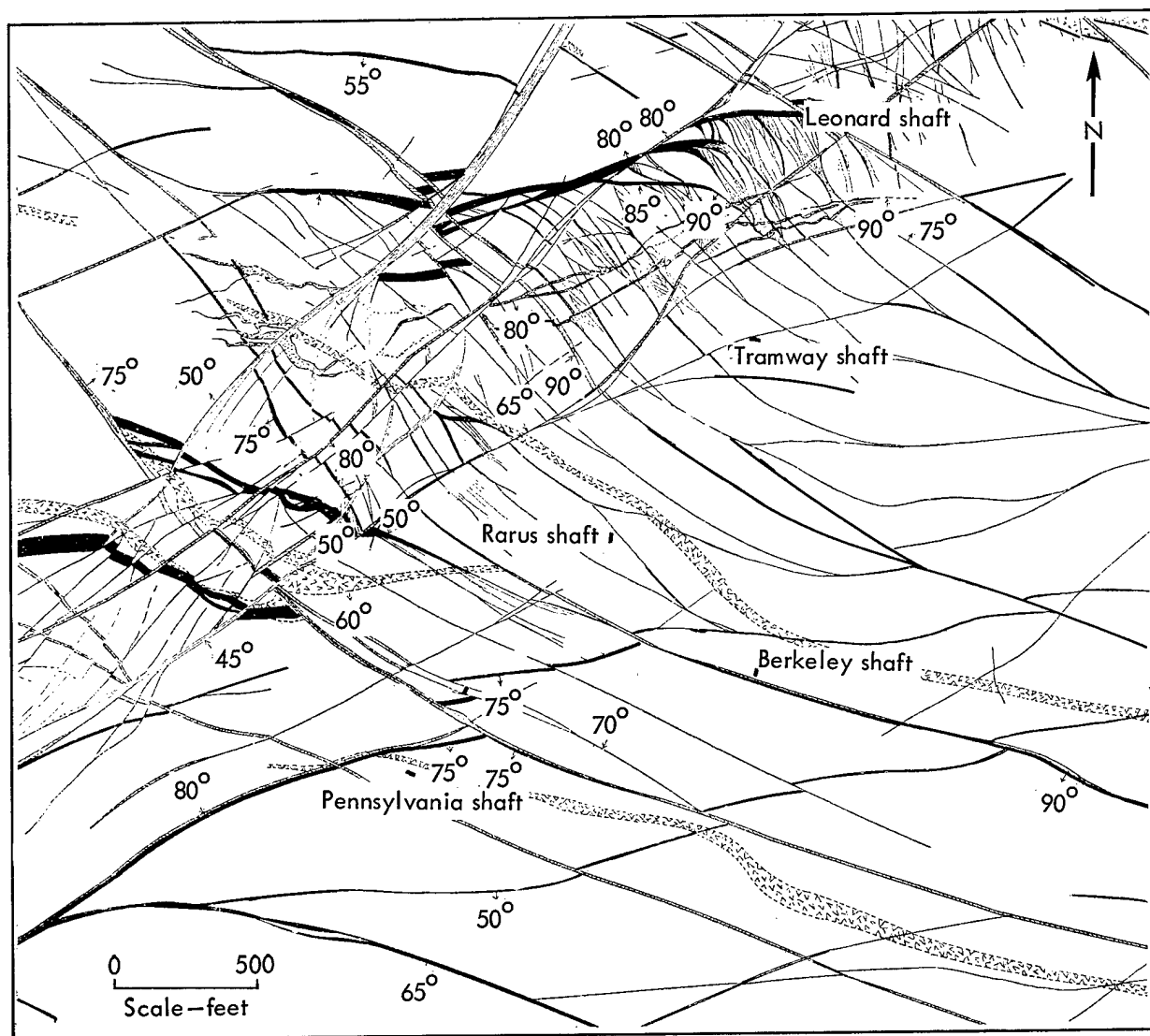


Figure A. 5 Eastern portion of Butte Mining District; showing pattern of veins and faults in the vicinity of the Berkeley Pit. (Adapted from Reference 112, Sales, 1914)

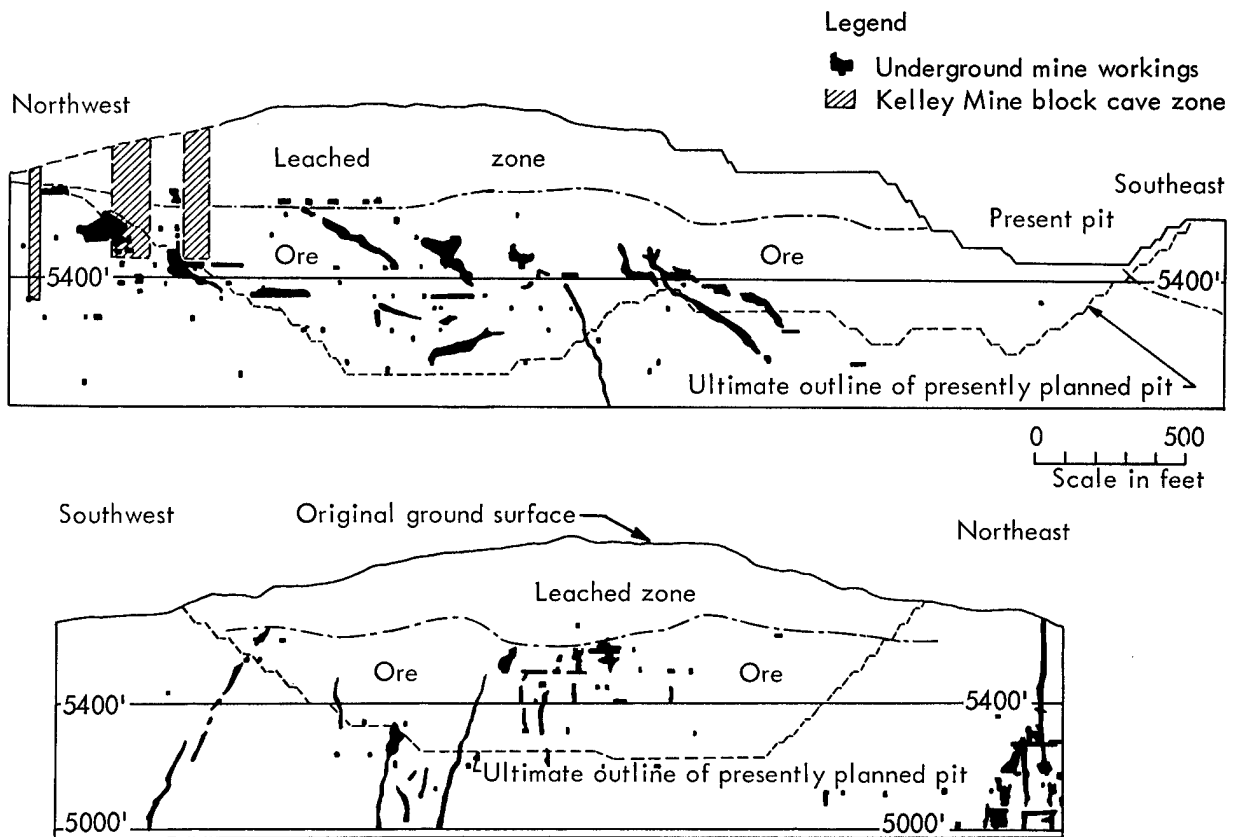


Figure A.6 Berkeley Pit; sections showing present and planned pit profile, original ground surface, and underground mine workings including block-caved rubble zones. (Adapted from Reference 74, McWilliams, 1958)

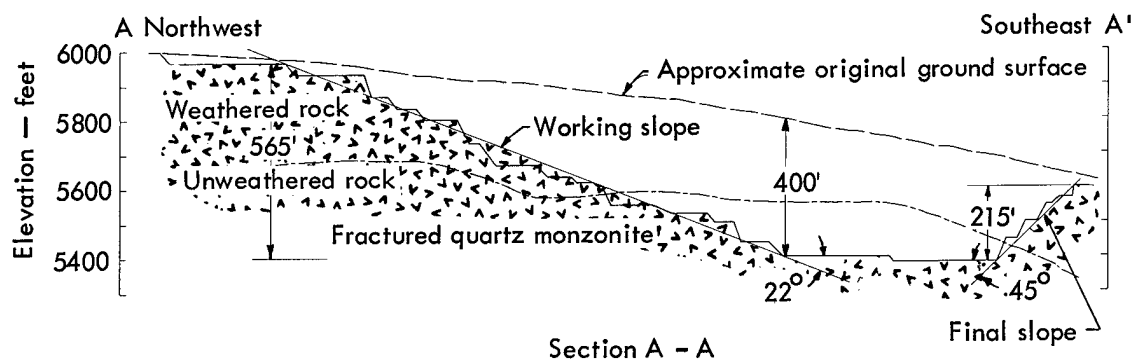
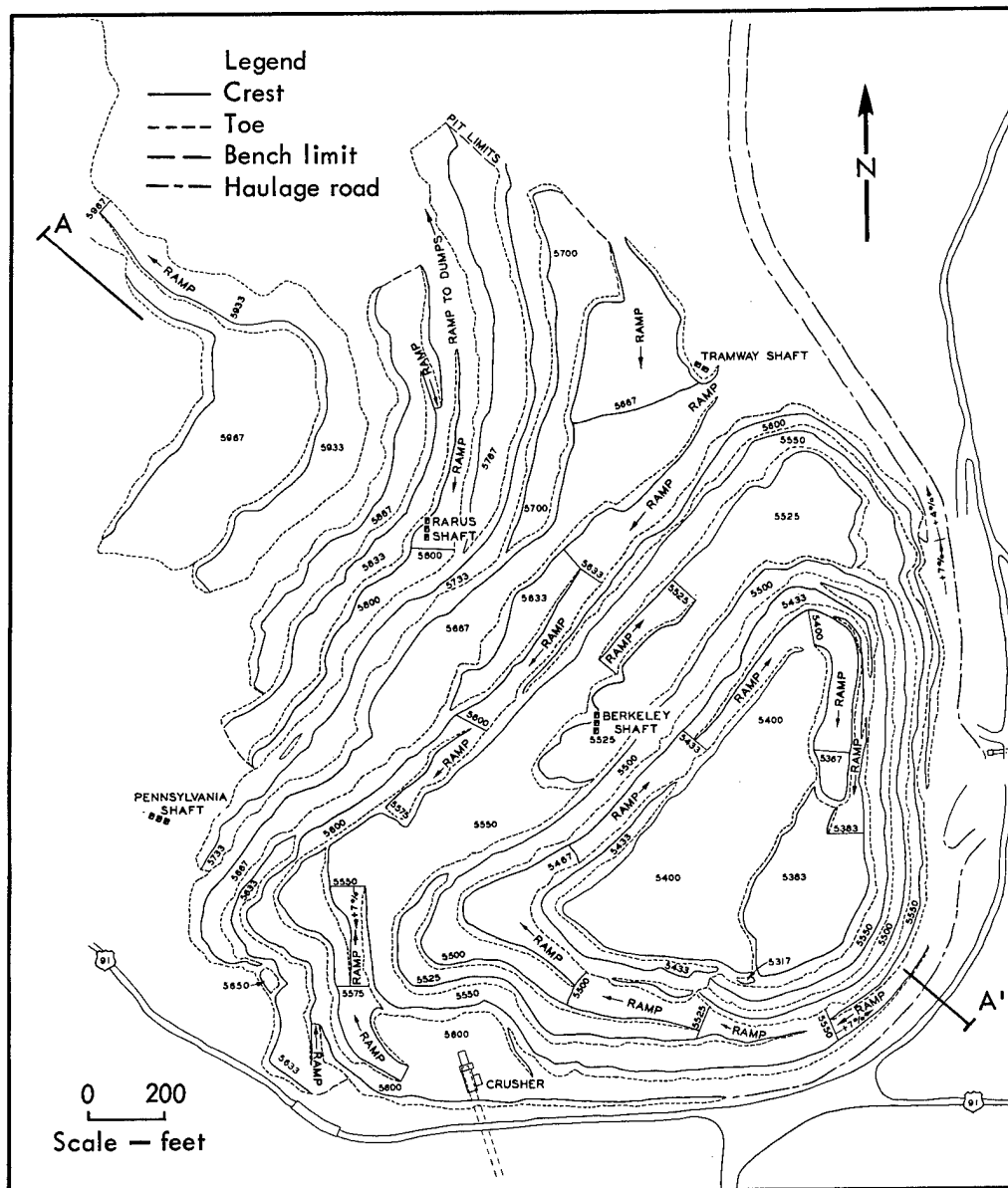


Figure A.7 Berkeley Pit; plan and generalized geological section. (Adapted from Reference 74, McWilliams, 1958)

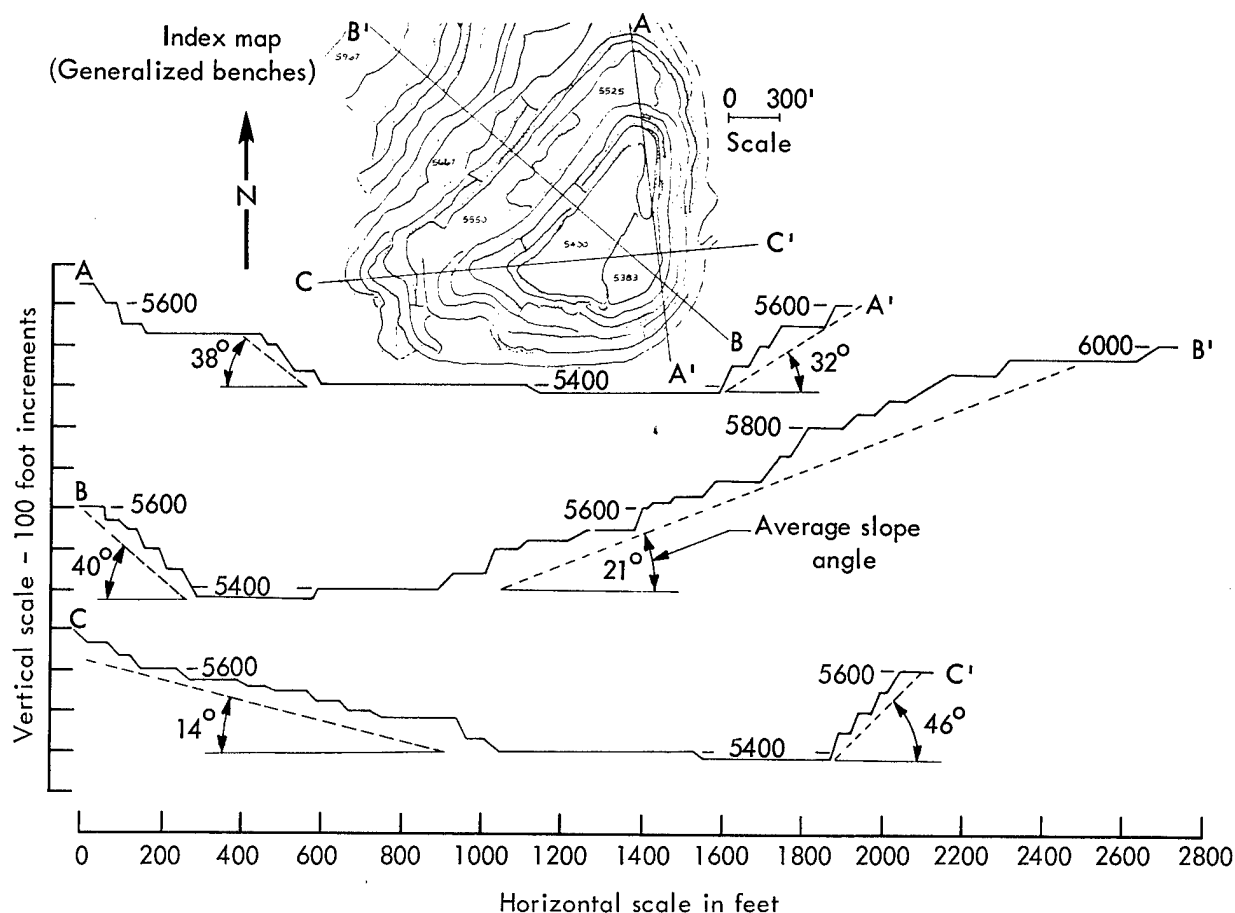


Figure A.8 Berkeley Pit; generalized plan and representative profiles. (Adapted from Reference 74, McWilliams, 1958)

The design of this pit has been influenced by a number of geological and non-geological factors, including the following:

- a. Configuration and tenor of the supergene ore zone
- b. Location of major railroads, highways, and urban structures
- c. Location of underground mines in the immediate vicinity of the pit
- d. Maximum grade of access and haulage roads
- e. Local topography – specifically, a sharp rise toward the northwest.

No significant slope failures occurred in the Berkeley Pit until quite recently, though localized zones of subsidence resulting from earlier underground block-caving operations necessitated some local slope modifications. Within these zones of subsidence maximum slopes are held to angles of 38° , in contrast to the overall design slope of 45° .

Slide movements of significant proportions (cf. Figure A.9) are reported to have occurred in the Berkeley Pit in the latter part of 1966. No reports are available, however, concerning the possible causes of this instability.

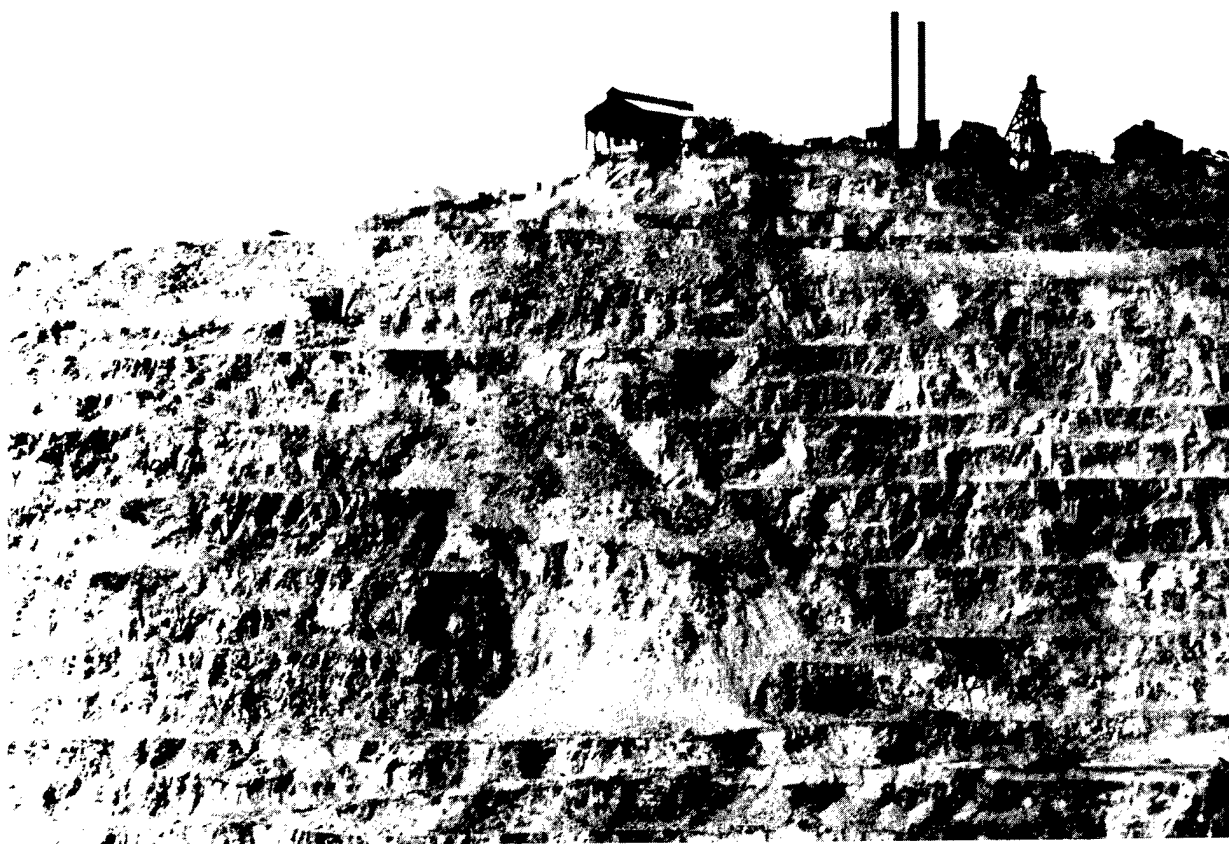


Figure A.9 Berkeley Pit; northwestern wall, showing slide area localized along a vein. Headframe on skyline shows location of Pennsylvania Shaft (cf. Figure A.7). (Reproduced by special permission of The Anaconda Company)

It may be of significance that the steepest pit walls are oriented at fairly high angles to the trend of major fault planes in the area, while the northwestern pit wall, which parallels the trend of several major faults, is cut to a substantially lower angle. It should be noted, however, that any of several other geological or non-geological factors may have governed this aspect of pit geometry.

BINGHAM CANYON MINE
BINGHAM CANYON, UTAH (5)

The Bingham Canyon Mine (Figure A.10) is located in the West Mountain mining district of north-central Utah, approximately 30 miles southwest of Salt Lake City. This district is situated within the central portion of the Oquirrh Mountains, the most easterly range of the Basin and Range physiographic province, just west of the



Figure A.10 Bingham Canyon Mine; view northeastward from southwestern rim.
(Reproduced by special permission of Bonneville News Company)

Wasatch Front. The climate of the area is semiarid, with an average annual rainfall of from 10 to 15 inches. Temperatures range from -20 to $+100^{\circ}\text{F}$. At present, activity within the district includes open pit mining of low-grade copper ore and underground mining of lead-zinc-silver ore.

The West Mountain Mining District was organized in 1863. Early interest was primarily in the lead-silver ores. Small amounts of copper ore were mined in the late 1860's, and copper was extracted as a by-product in the 1870's and 1880's. Underground mining of high-grade copper ore started in 1897 and continued until 1947. The low-grade copper deposits of the region were first investigated in 1887. Open pit mining of these deposits commenced in 1906 and has continued to the present.

The major open pit mine of the district is situated in Bingham Canyon, a gorge up to 3,000-feet deep. Elevations within the area, prior to pit excavation ranged from approximately 6,100 to 8,700 feet. Natural slopes within the canyon average about 26°.

Bedrock formations at the Bingham Canyon Mine include interbedded quartzite and limestone which has been intruded by stocks, sills, and dikes of monzonite porphyry (Figure A.11). Major sedimentary structures of the area include a series of parallel

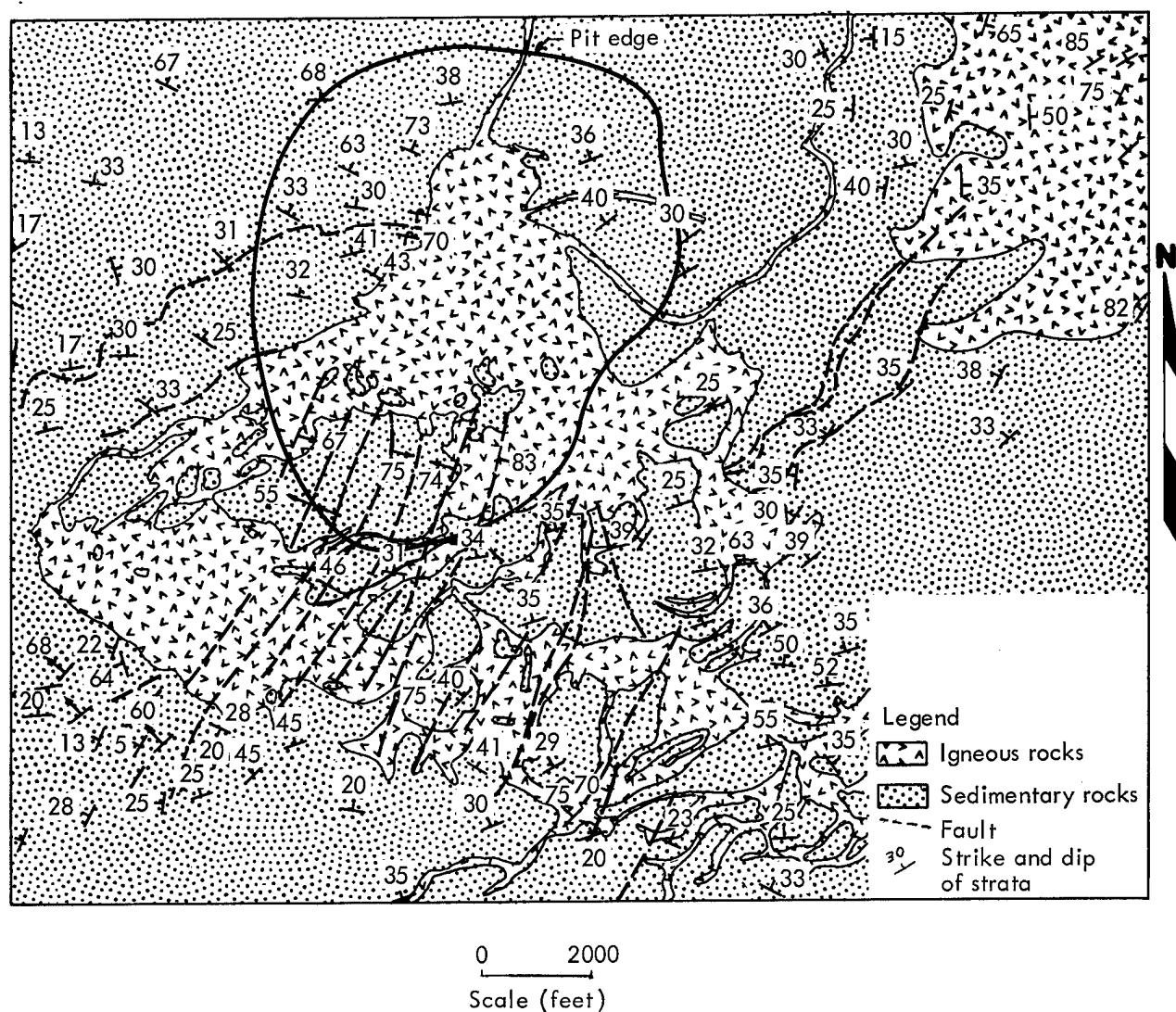


Figure A.11 Bingham Canyon Area; generalized geological map. (Adapted from Reference 150, Utah Geol. Soc., 1961)

plunging folds which are cut by numerous high angle and thrust faults. The location and configuration of intrusive bodies is closely related to the local fold and fault patterns.

Ore-grade mineralization at the Bingham Canyon Mine is confined to a roughly triangular zone with sides ranging from 5,000 to 7,000 feet in length. This zone extends

to a depth of about 2,000 feet, overlain by an average of 115 feet of barren leached material. The ore body is largely localized within the Bingham stock, a monzonite porphyry intrusive, although some ore-grade mineralization is also found in adjacent portions of the limestone and quartzite formations.

The Bingham Canyon Pit has been largely excavated in the igneous rocks of the Bingham stock. This rock is highly altered and permeated by a network of small diversely oriented fractures spaced only a few inches apart. In general, the bedrock of the pit is rather soft and breaks readily along these established fracture planes.

The pit is roughly circular in plan, with diameters ranging from approximately 7,000 to 12,000 feet. Maximum relief within the pit is about 2,300 feet. Average slope angles range from 19 to 26°. Continuous slopes of 24° range up to approximately 6,000 feet in length (Figure A.12).

Slope stability problems of both major and minor proportions have developed throughout the history of the Bingham Pit. These problems are reflected in the discrepancy between the present slope configuration, as described above, and as shown in Figure A.13, and the initial design criteria which called for average slopes of 40°, with a maximum declivity of 50° and a minimum of 35°. The slides that occurred have been attributed, with few exceptions, to adverse structural conditions. This is particularly true along the western and southwestern walls of the pit, where the presence of several major faults has led to slope failures of greatest frequency and severity.

In 1956, a slide of major proportions occurred on the west side of the pit. It involved approximately 1,250 vertical feet of the pit face. Movement was rapid and was preceded only by minor cracking and surface moisture seeps. A detailed analysis of this slide revealed that the movement had occurred along existing fault planes. Instability was apparently caused by the hydrostatic pressure induced by excessive ground water. A generalized section through the slide is shown in Figure A.14.

References 8, 15, 67, 89, 102, 122, 141, 144, 150, 179 (cf. Figure 3.7)

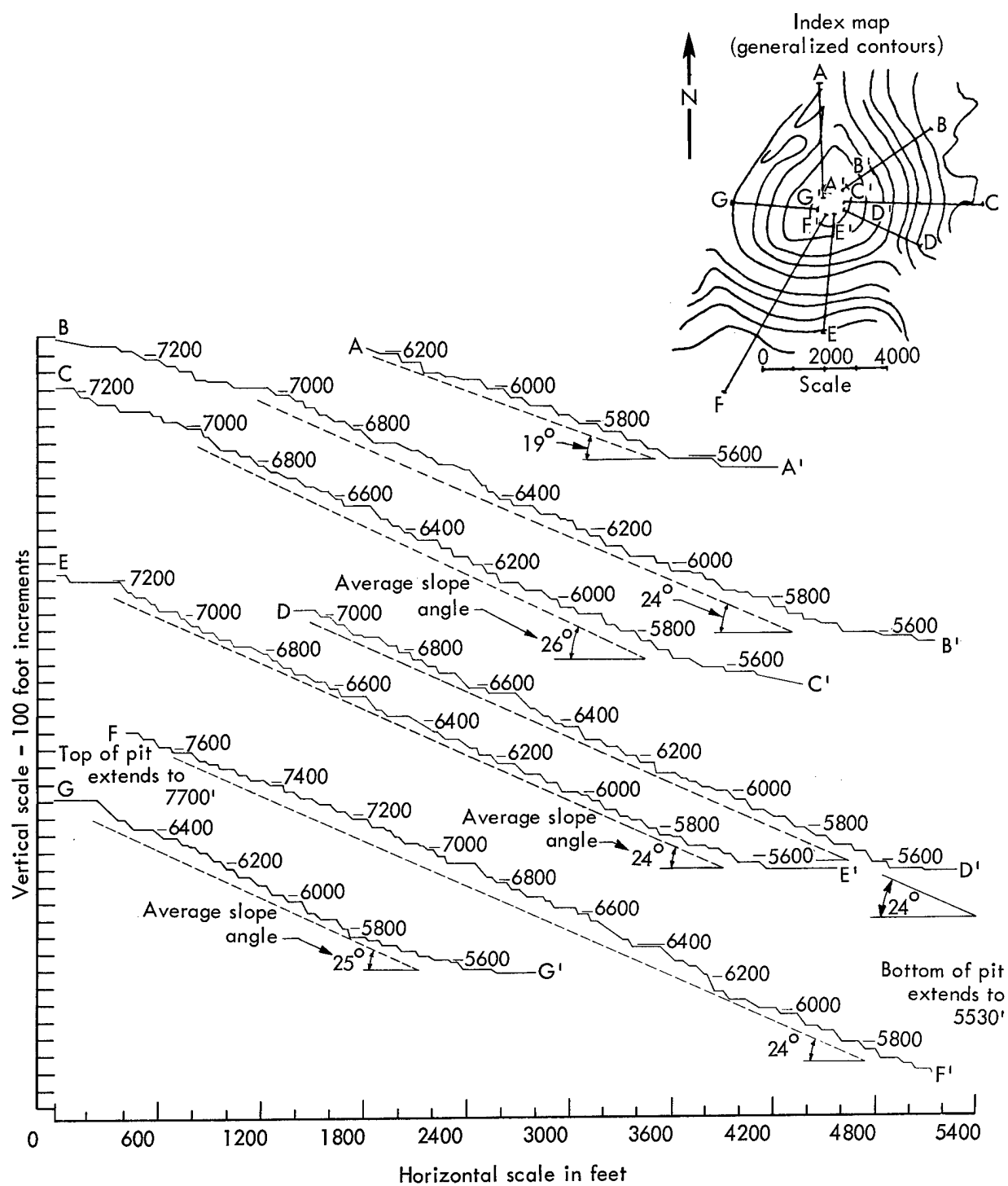
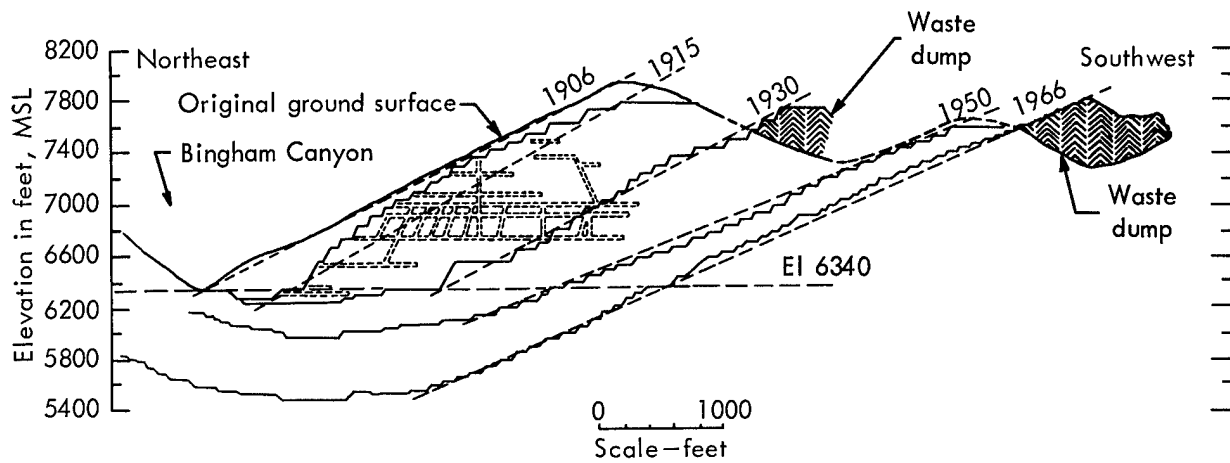


Figure A.12 Bingham Canyon Mine; generalized plan and representative profiles.
(Adapted from data contributed by the Kennecott Copper Corporation)



Year	Slope dimensions			
	Depth of Excavation D	Slope Height H	Slope Angle	Rate of Excavation
	ft	ft	Deg	Tons per day
1906	0	1590	21	60,000
1915	370	1480	28	120,000
1930	950	1470	27	210,000
1950	1400	1590	21	270,000
1966	1750	2280	23	330,000

Figure A.13 Bingham Canyon Mine; representative profiles showing progress of excavation. (Adapted from data contributed by the Kennecott Copper Corporation)

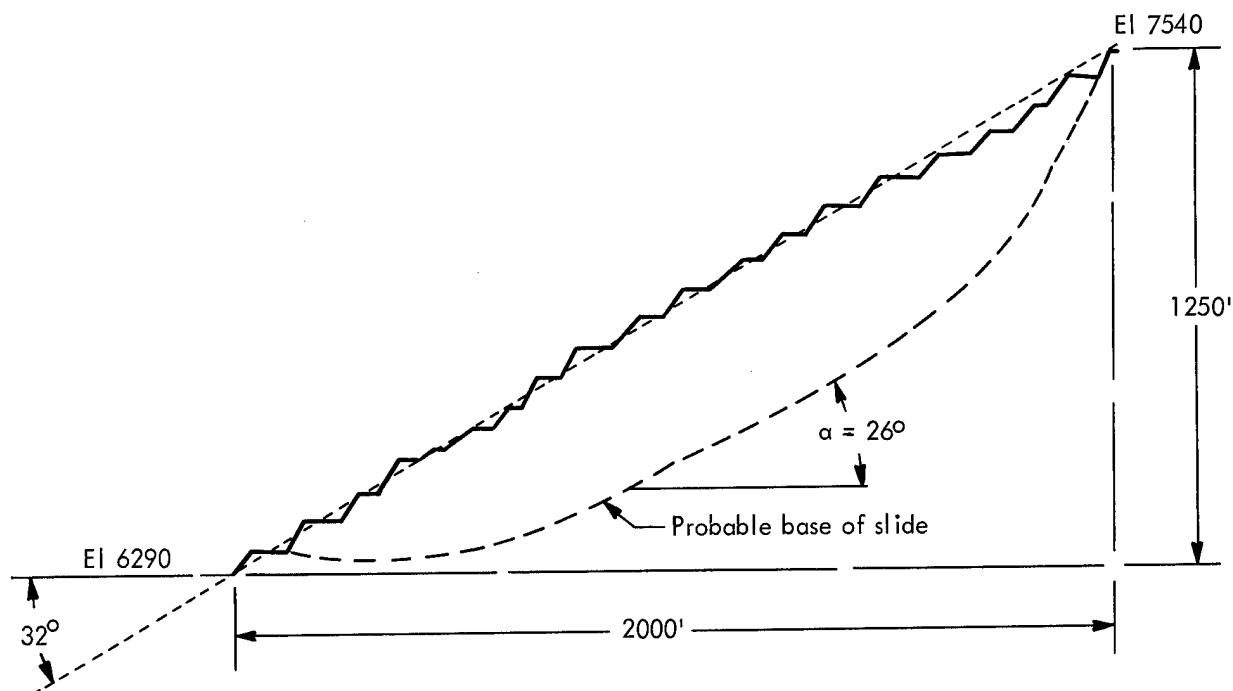


Figure A.14 Bingham Canyon Mine; section through major slide in western pit wall. (Adapted from Reference 144, Wilson, 1959)

BORON PIT BORON, CALIFORNIA (6)

The Boron Pit is located in the northwestern part of the Mojave Desert of southern California, about 100 miles northeast of Los Angeles, and immediately north of the town of Boron. The climate of the region is arid, with only about 8 inches of rainfall annually. High temperatures prevail during the summer months.

The ore body is roughly lenticular in shape and ranges in thickness from a few feet at its outer limits to several hundred feet in its central portion (Figure A.15). The ore body, consisting of massive borax with varying amounts of intercalated claystone, is overlain by poorly to moderately consolidated arkose and conglomerate. Folding and faulting have produced local deformation of the sedimentary strata (Figure A.16). The predominant faults are normal with dips ranging from 30° to vertical.

Open pit mining operations at Boron were initiated in 1956. Prior to that time, open stopes remaining from earlier underground mining were backfilled with sandy overburden.

The Boron Pit is roughly rectangular in plan, although this shape is somewhat distorted by a bulge in the central part of the south slope (Figure A.17). The pit measures about 3,000 feet in an E-W direction and 2,000 feet in a N-S direction. The

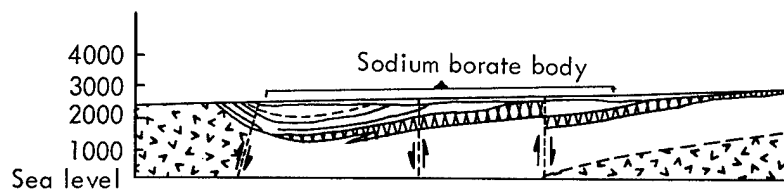
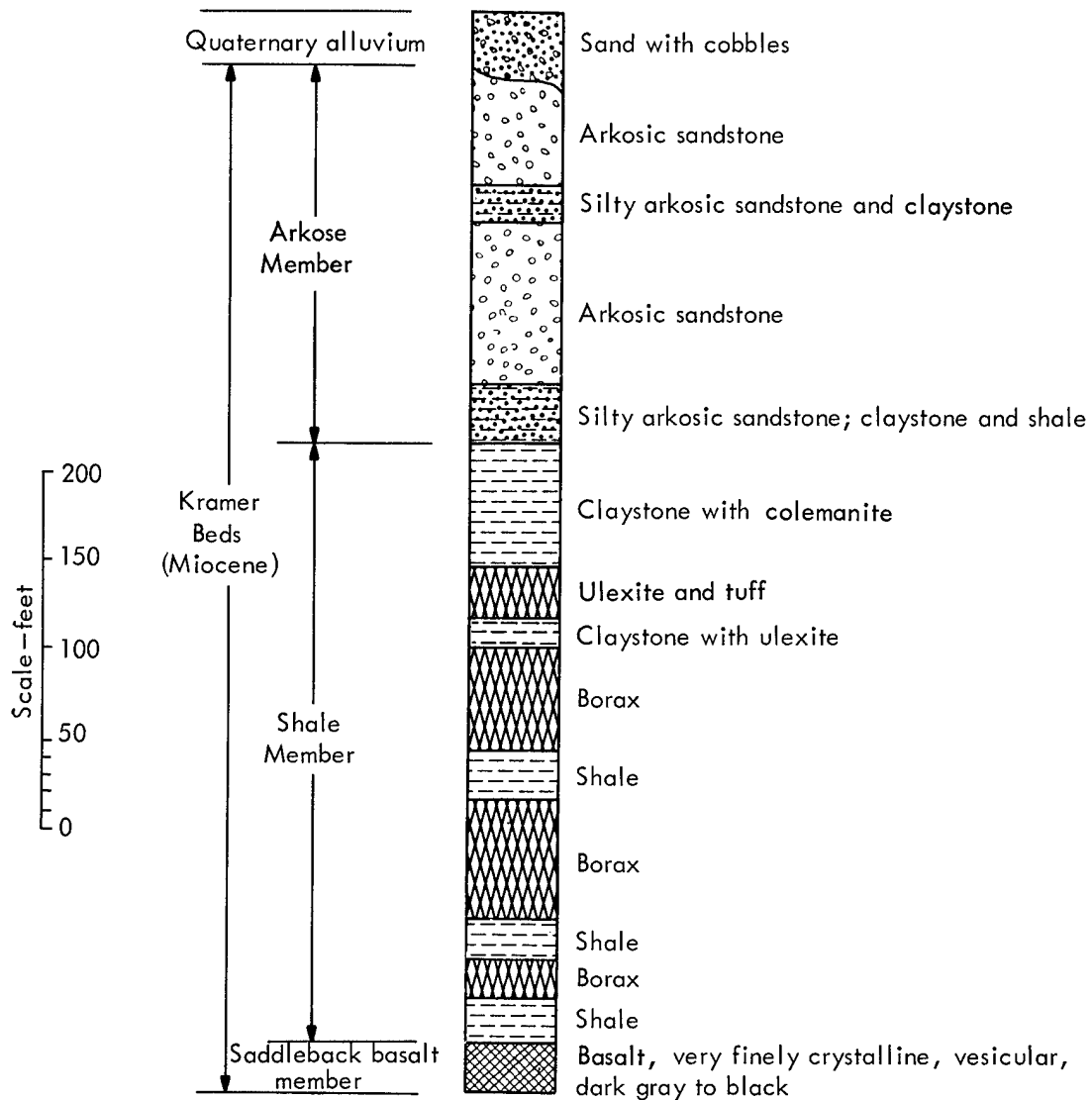


Figure A.15 Boron Pit; stratigraphic column (Adapted from Second Symposium on Salt by special permission of the Northern Ohio Geological Society) and generalized geological section (Adapted from reference 9, Bates, 1960)

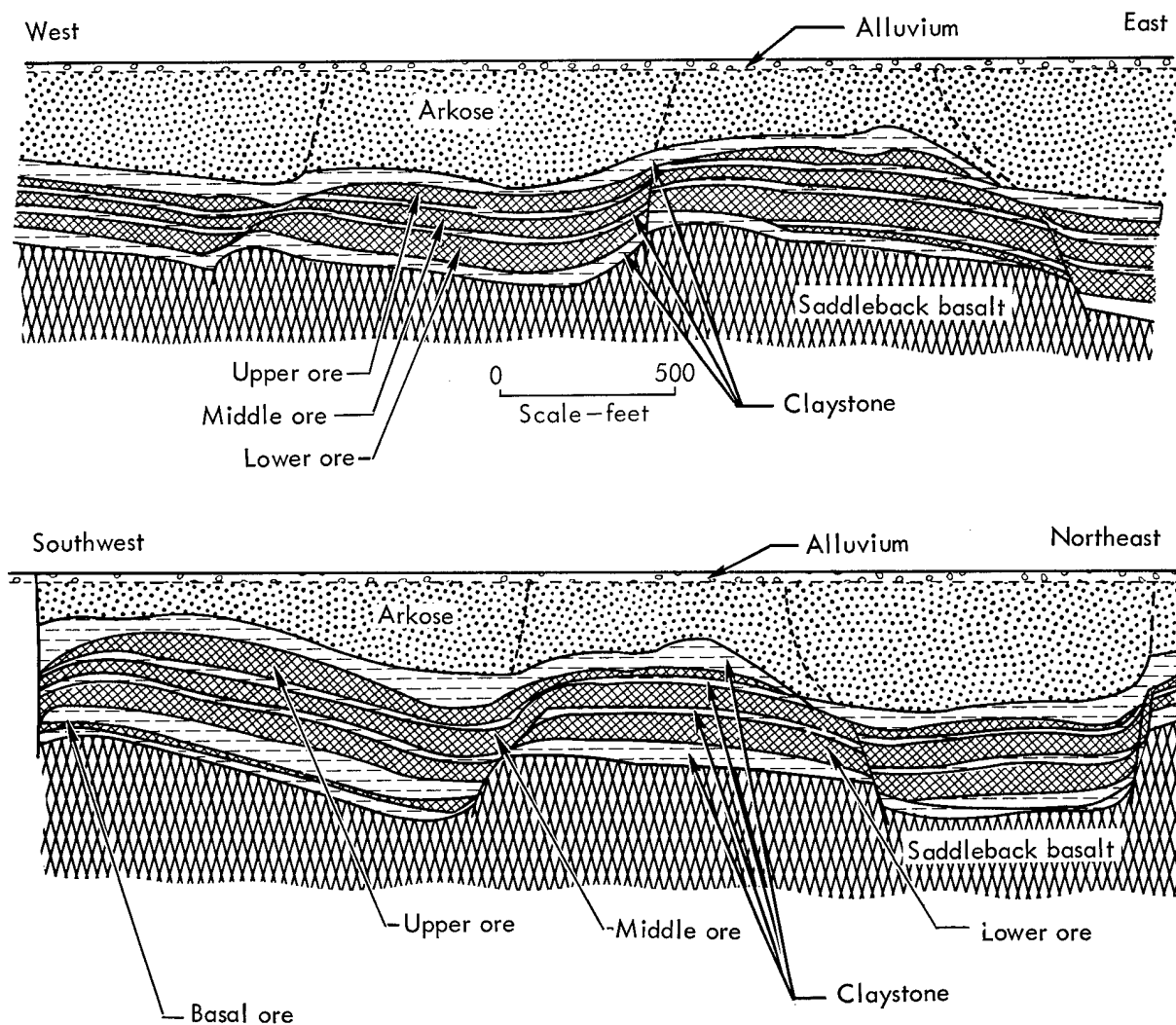


Figure A.16 Boron Pit; representative geological sections. (Adapted from Second Symposium on Salt by special permission of the Northern Ohio Geological Society)

steepest slopes (central part of the north and south slopes) in the pit are about 30° and are about 700 to 800 feet in length (Figure A.18). Pit walls are broken at vertical intervals of 50 feet by benches averaging about 40 feet in width. Individual benches are slopes at about 63° .

Several significant slides and slumps have occurred in the southern and eastern portions of the pit. The first major slide was noted in the southeastern sector in June 1962, at which time the depth of excavation was about 150 feet. About 250,000 tons were involved in the sliding mass. Vertical heaving of 10 to 12 feet occurred at the toe of the slide. Bedding planes in this area dipped toward the pit at angles of 18 to 22° ,

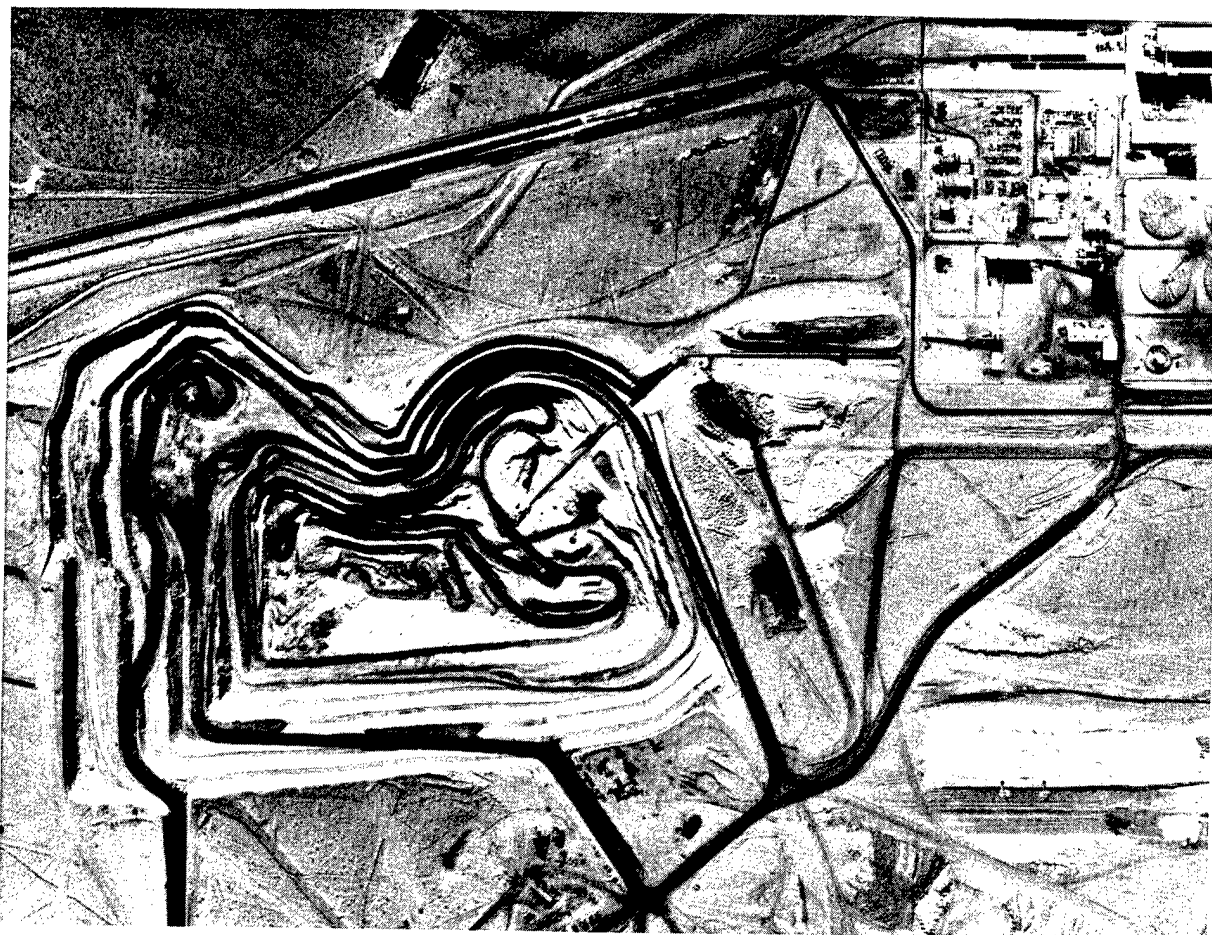


Figure A.17 Boron Pit; vertical aerial view. (Contributed by U. S. Borax and Chemical Corporation)

and the base of the sliding mass was marked by a 2-inch seam of bentonitic clay. In June 1964, a second slide occurred in this same general area, with cracks up to 150 feet in length developing as far back as 30 feet from the rim of the pit. By November 1964, cracks could be observed as far as 300 feet back from the rim, and these cracks continued to open as the pit was enlarged and deepened. The base of this sliding mass was defined by a fault dipping toward the pit at 45° . Bedding planes within the sliding mass also dipped toward the pit, but at a lower angle.

In 1964, geophones were installed in drill holes some distance behind the slide zone, and precise survey measurements were initiated in order to monitor slope movement. A definite correlation was noted between geophone noise levels and rates of slope adjustment. When about 50 feet of material was removed from the top of the moving mass in April 1965, the geophone noise level decreased and no further movement was observed for about a year. Since this unloading, however, the pit has been deepened about 100 feet in this sector, and the mass has again started to move.

Another major slide occurred on the east side of the pit in April 1963. At that time the overall pit slope in this area was 34° . The movement took place in about

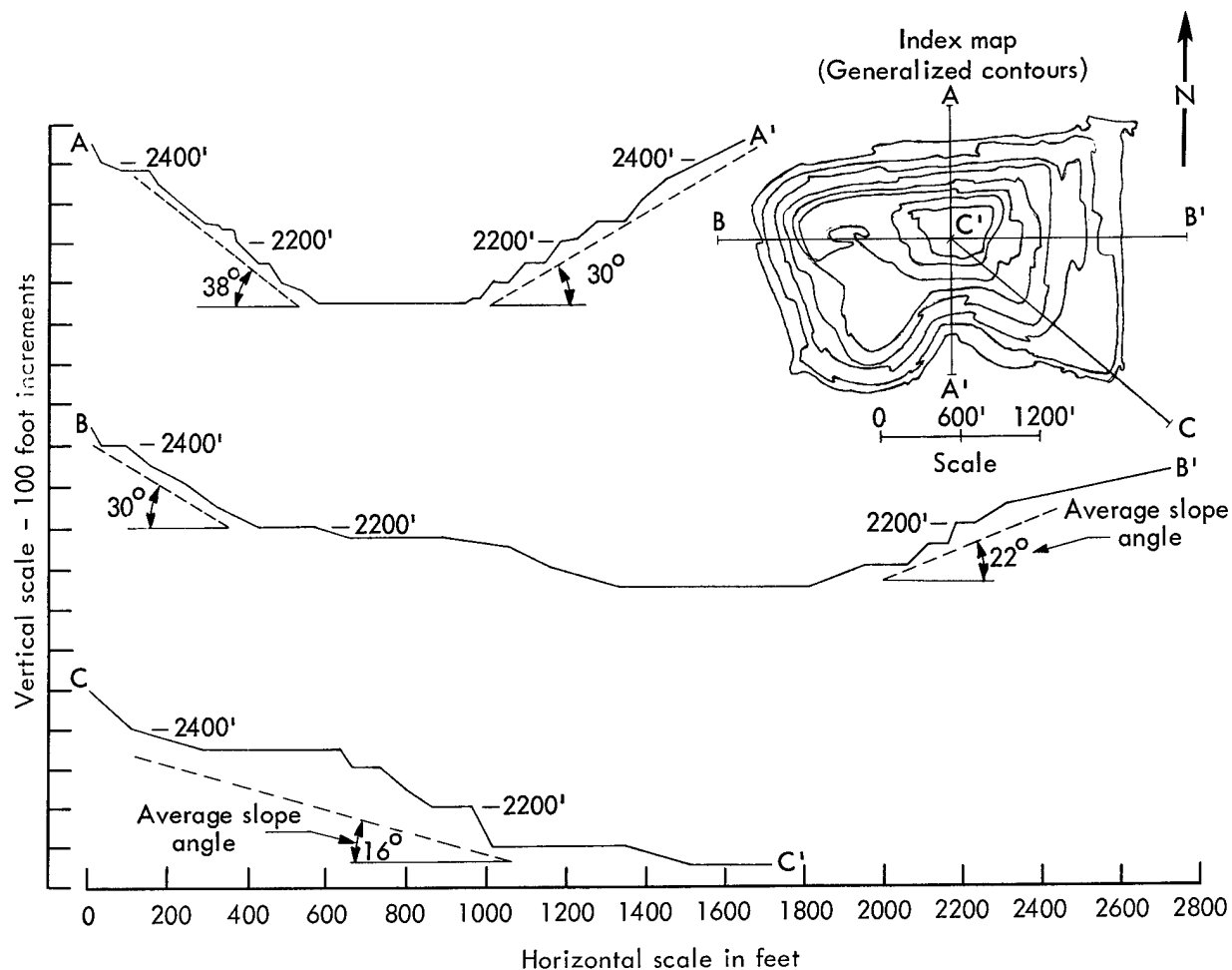


Figure A.18 Boron Pit; generalized plan and representative profiles. (Adapted from data contributed by U. S. Borax and Chemical Corporation)

5 minutes, with the top of the slide slumping about 100 feet and the mass coming to rest at an angle of about 18° . The base of the slide was marked by clay and sand strata which dipped toward the pit at about 20° . In July 1963, about 50 tons of explosives were detonated in holes behind the slide in order to move the entire mass into the pit, but this effort proved unsuccessful.

CASTLE DOME MINE
MIAMI, ARIZONA (9)

This open pit copper mine (Figure A.19) was excavated in a porphyritic quartz monzonite intrusive which has been subjected to argillic, sericitic, and propylitic metasomatism.



Figure A.19 Castle Dome Mine; oblique aerial view. (Contributed by Miami Copper Company)

When mining operations were terminated at Castle Dome in 1952, the pit was 760 feet deep, and the benched walls were trimmed to a final slope of 45° (Figure A.20). As of 1957, these 45° slopes had remained stable, although some minor spalling had occurred.

The ore-bearing quartz monzonite occupies a horst structure bounded by steeply dipping normal faults which trend NNW-SSE. The most prominent structure exposed within the pit is the Dome fault, which nearly bisects the excavation. Many subsidiary faults striking NW-SE and dipping northeast are also noted in the mine area.

Two major joint sets are developed in the mineralized intrusive. The most prominent of these trends ENE-WSW, and dips steeply southward. The other trends

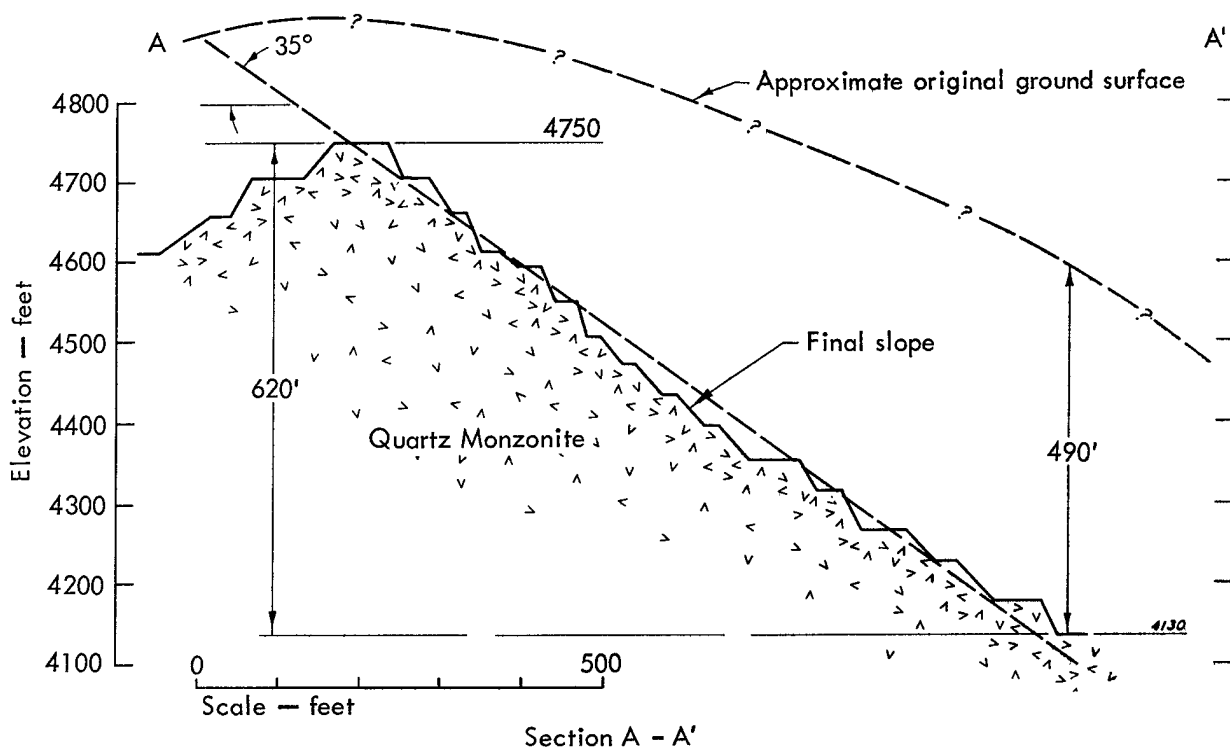
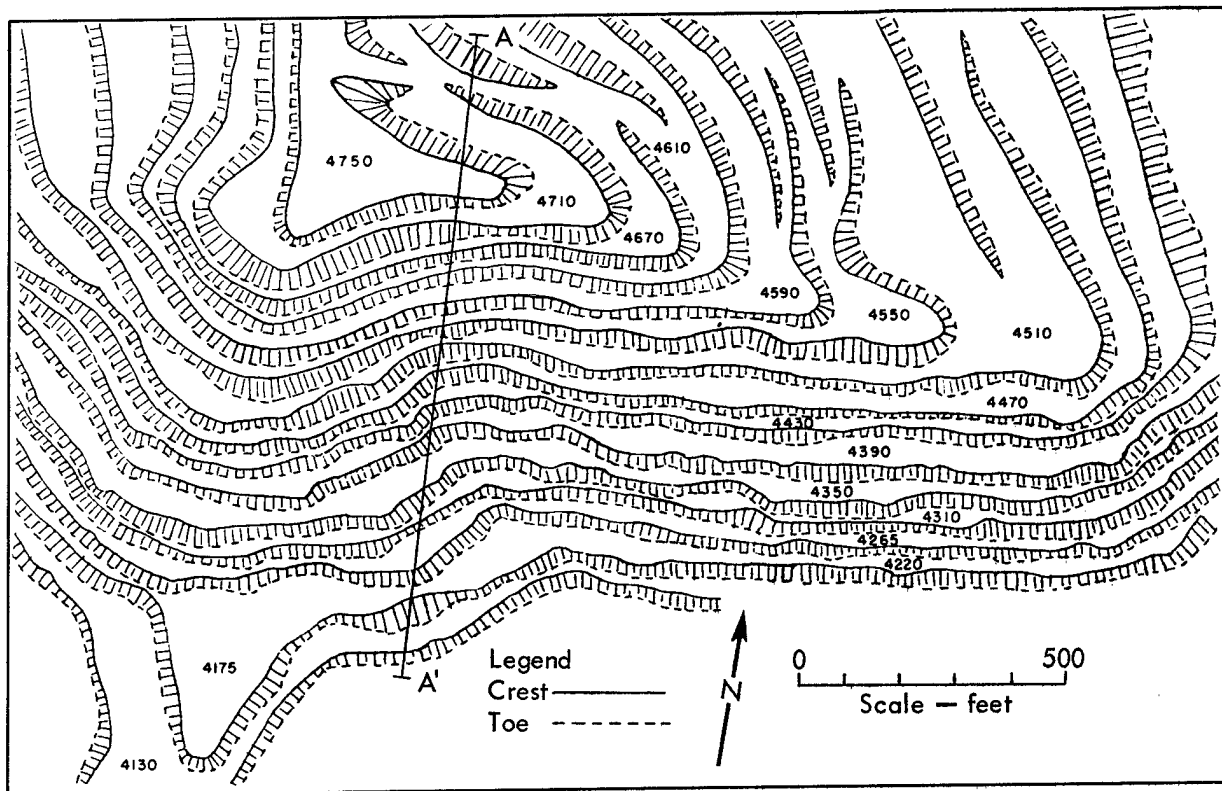
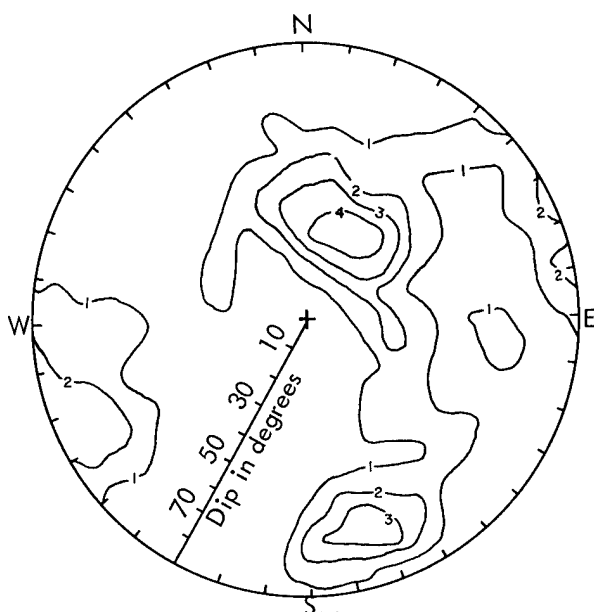


Figure A.20 Castle Dome Mine; plan and representative profile. (Adapted from Reference 52, Hardwick and Stover, 1960)

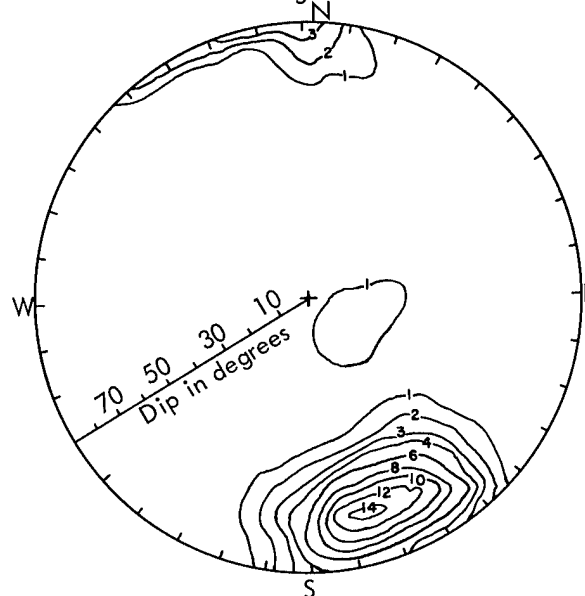
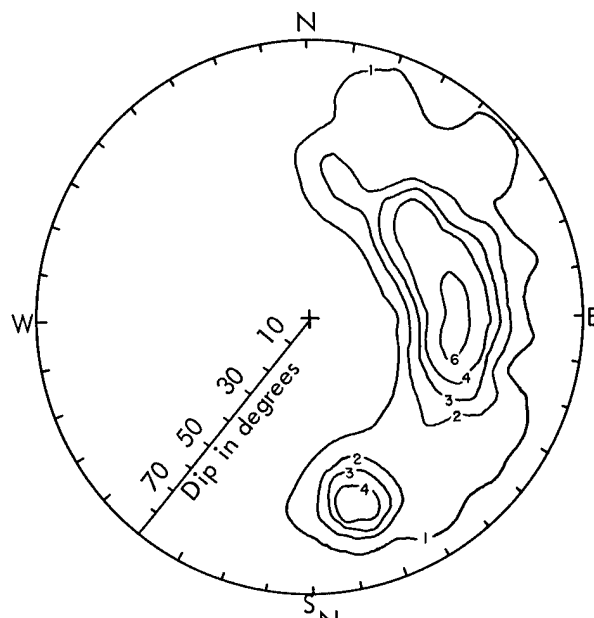
NNW-SSE and dips to the northeast at moderate angles (Figure A.21). In some bench faces closely spaced joints of the second set impart a sheeted appearance to the rock. Jointing is generally most intense in the southeastern portions of the pit.

The water table in the area lies well below the floor of the pit.

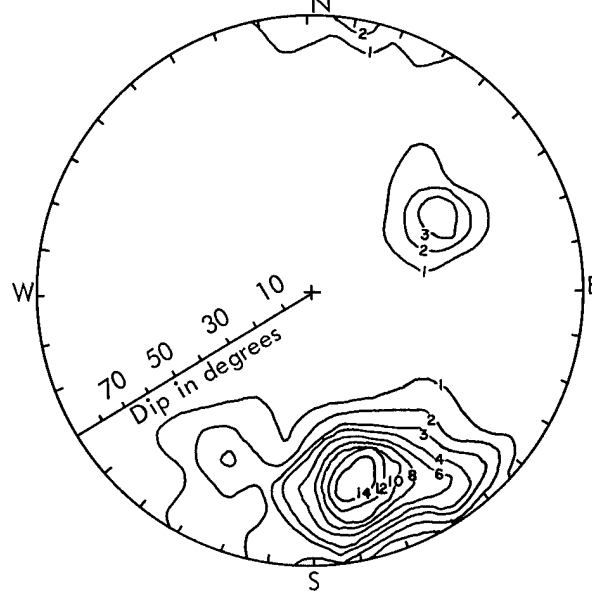
A. Attitude of joints west of Dome Fault; percentage of 745 joints



B. Attitude of joints east of Dome Fault; percentage of 1332 joints



C. Attitude of veins west of Dome Fault; percentage of 690 veins



D. Attitude of veins east of Dome Fault; percentage of 655 veins

Figure A.21 Castle Dome Mine; stereograms of vein and joint orientations. (Adapted from Reference 93, Peterson, Gilbert, and Quick)(1951)

CHAMBISHI MINE
CHAMBISHI, ZAMBIA (10)

This open pit copper mine has been developed in a sequence of folded metasedimentary rocks. The pit has been excavated to a maximum depth of about 400 feet in an area of initially level topography (Figure A.22).

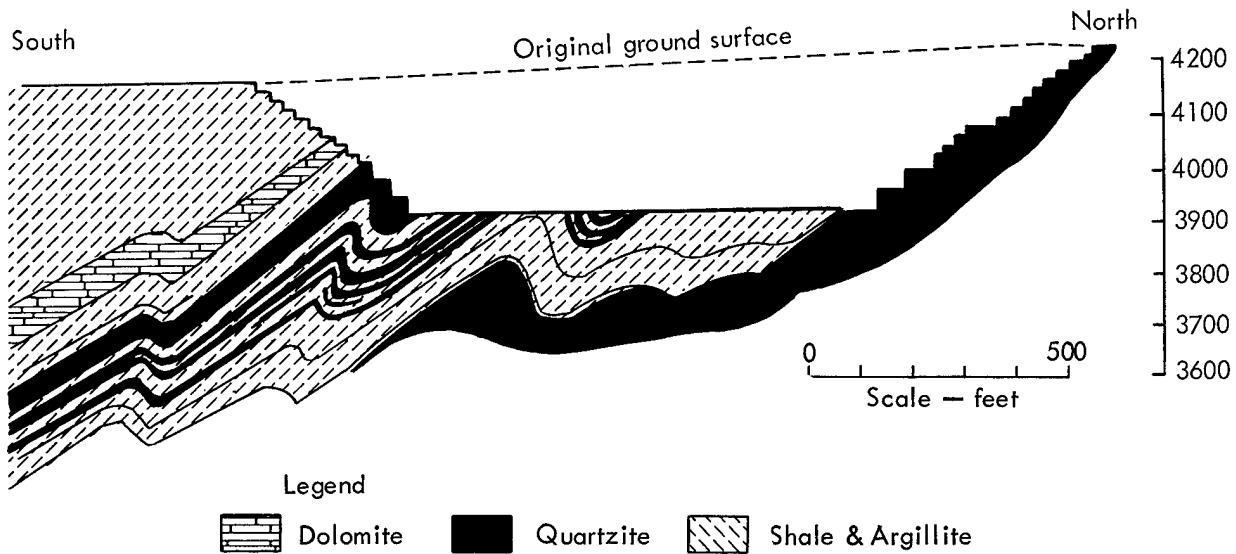


Figure A.22 Chambishi Mine; representative cross section. (Adapted from data contributed by Roan Selection Trust, Ltd.)

Average declivity of the benched pit walls varied from about 45° in the lower one-third of the pit to about 30° in the upper two-thirds.

Despite a relatively high water table (less than 100 feet below the original topographic surface), no serious ground-water-related slope stability problems have been reported. A photograph provided by the operating company (Figure A.23) indicates that the eastern wall of the pit (cut approximately perpendicular to the strike of the folded bedrock units) is stable. Two slide areas, however, can be seen in the upper portions of the southern wall (cut approximately parallel to the strike of the folded bedrock units).

References 75, 189 (cf. Figure 2.8).



Figure A.23 Chambishi Mine; oblique aerial view, looking eastward. (Contributed by Roan Selection Trust, Ltd.)

DAY-LOMA AND FRAZIER-LAMAC PITS FREMONT AND NATRONA COUNTIES, WYOMING (17, 23)

These open pit uranium mines are located in the Gas Hills Mining District along the southern edge of the Wind River Basin. Relief in this district is moderate, with a mean elevation of 6,600 feet above sea level. The land surface slopes gently northward toward the Wind River Basin, and rises sharply southward to the Beaver Rim escarpment which has an average elevation of 7,000 feet. The region is semi-arid, with an average annual precipitation of 12 inches. Regional drainage channels trend northward toward the Wind River Basin, but carry water only during the spring or during an occasional heavy rainfall.

The ground water level is, on the average, about 150 feet below the surface in this area, though perched water tables are frequently encountered at higher levels. The water table is steeply depressed in the vicinity of the pits, except in zones of unusually permeable bedrock.

The Frazier-Lamac and Day-Loma Pit complexes (Figures A.24 and A.25) have been excavated to depths of approximately 300 feet in clastic sedimentary rocks, including fine- to coarse-grained cross-bedded arkosic sandstones, bentonitic siltstones, and boulder conglomerates. The sandstones are generally friable, but are cemented by calcite and/or pyrite in the immediate vicinity of uranium ore bodies. The other bed-rock units are comparatively well consolidated, but are sufficiently weak (except for some portions of the conglomerate) to permit their removal without blasting.



Figure A.24 Day-Loma Pit; vertical aerial view.
(Contributed by Western Nuclear, Inc.)



Figure A.25 Frazier-Lamar Pit; vertical aerial view.
(Contributed by Western Nuclear, Inc.)

Minor E-W trending faults and clay-filled subsidiary fractures are numerous in the Gas Hills area.

Walls of the Frazier-Lamac Pit are broken by benches 15 to 70 feet in width at 20- to 80-foot vertical intervals. Benches are sloped at 65 to 80° to produce average overall slopes of 40 to 55°.

References 35, 70, 195

EAST JERSEY AND JERSEY PITS ASHCROFT, BRITISH COLUMBIA (19, 27)

These open pit copper mines are located at an elevation of about 5,000 feet in a heavily forested region of gently rolling hills near Ashcroft, British Columbia. The pits have been excavated in a series of intermediate plutonic and volcanic rocks which are transected by numerous faults and porphyry dikes (Figure A.26).

The East Jersey Pit is oval in plan (elongated on a N-S axis), and measures about 1,000 feet in length and 500 feet in width. The pit has been excavated to a depth of 400 feet, and is no longer in operation.

Bedrock structure in the vicinity of the East Jersey Pit is dominated by two major fault systems. The dominant system strikes NNE-SSW and dips at high angles to the east or west; while a subsidiary system strikes ENE-WSW and dips steeply to the north or south (Figure A.27).

The walls of this pit are broken at vertical intervals of about 60 feet by benches averaging about 20 feet in width. Bench faces are sloped at 70°, and the average overall slope of pit walls is about 50°.

Operation of the East Jersey Pit was halted as a result of a massive slope failure which involved most of the eastern wall of the pit, and which covered the pit floor with 500,000 to 1,000,000 tons of broken wall rock. The zone of failure is oriented approximately parallel to the strike of the major structural (fault) system; and it is believed (according to a company geologist) that the immediate cause of the failure was an accumulation of runoff water in a swampy area near the southeast corner of the pit, and the percolation of this water into the underlying bedrock which was extensively fractured and jointed.

The newer Jersey Pit, which is being developed just to the west of the East Jersey Pit (Figure A.28), encounters substantially similar bedrock lithology and structure. Thus far, this pit has been excavated to a maximum depth of 480 feet, utilizing irregularly benched walls with a maximum overall declivity of about 40°. The mine geologist reports that

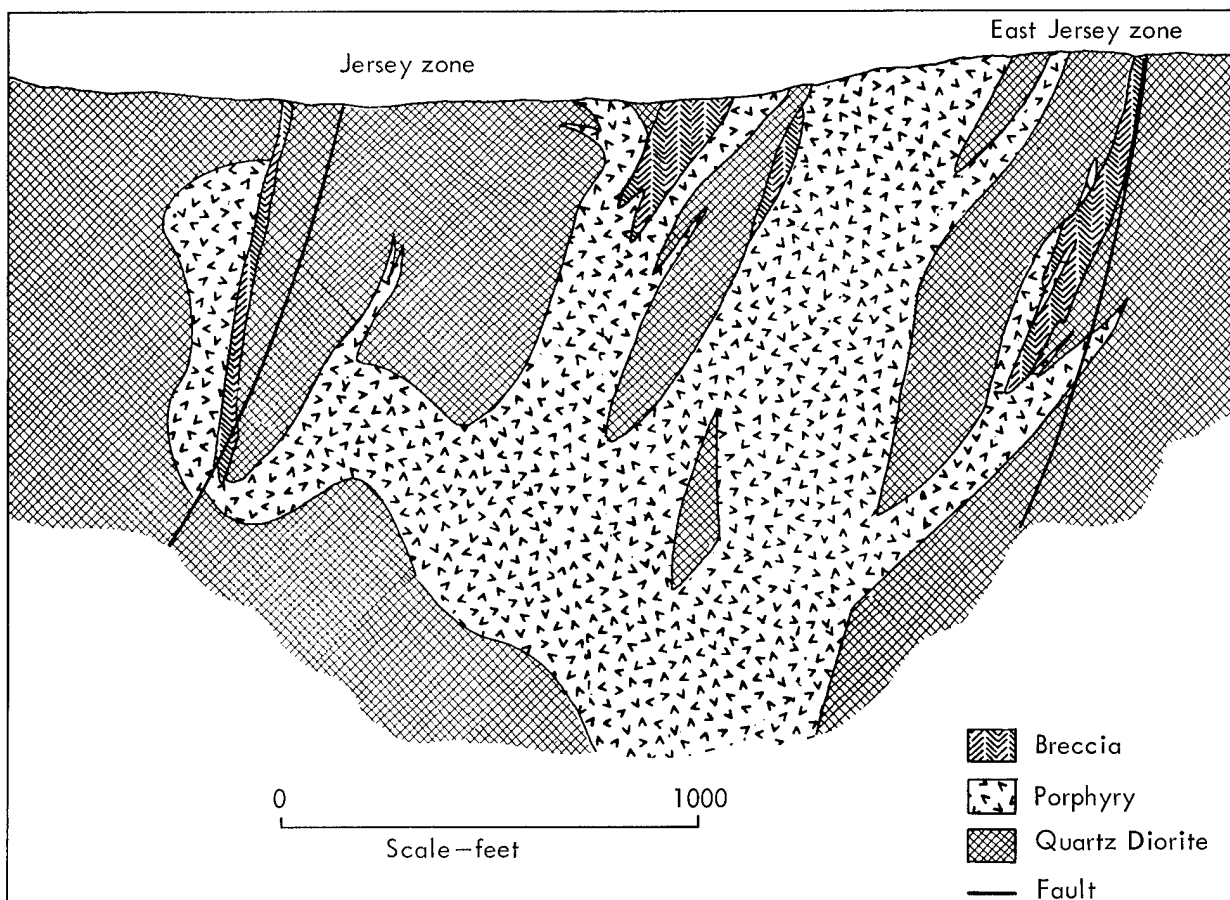
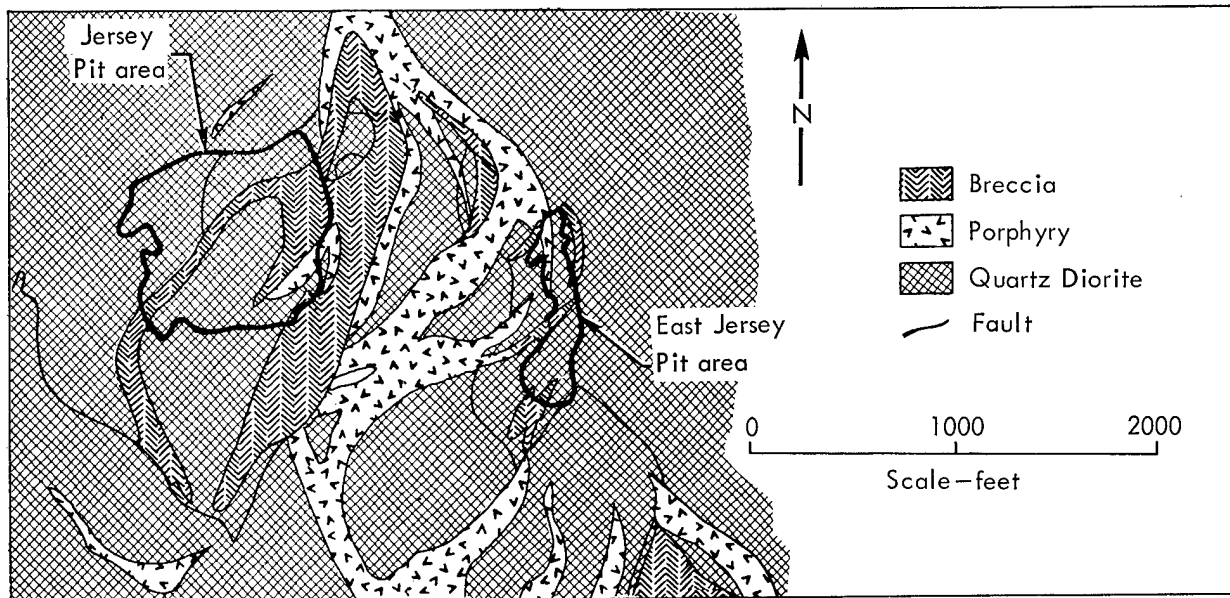


Figure A.26 Generalized geological map and section; vicinity of East Jersey and Jersey Pits. (Adapted from Reference 22, Carr, 1960)

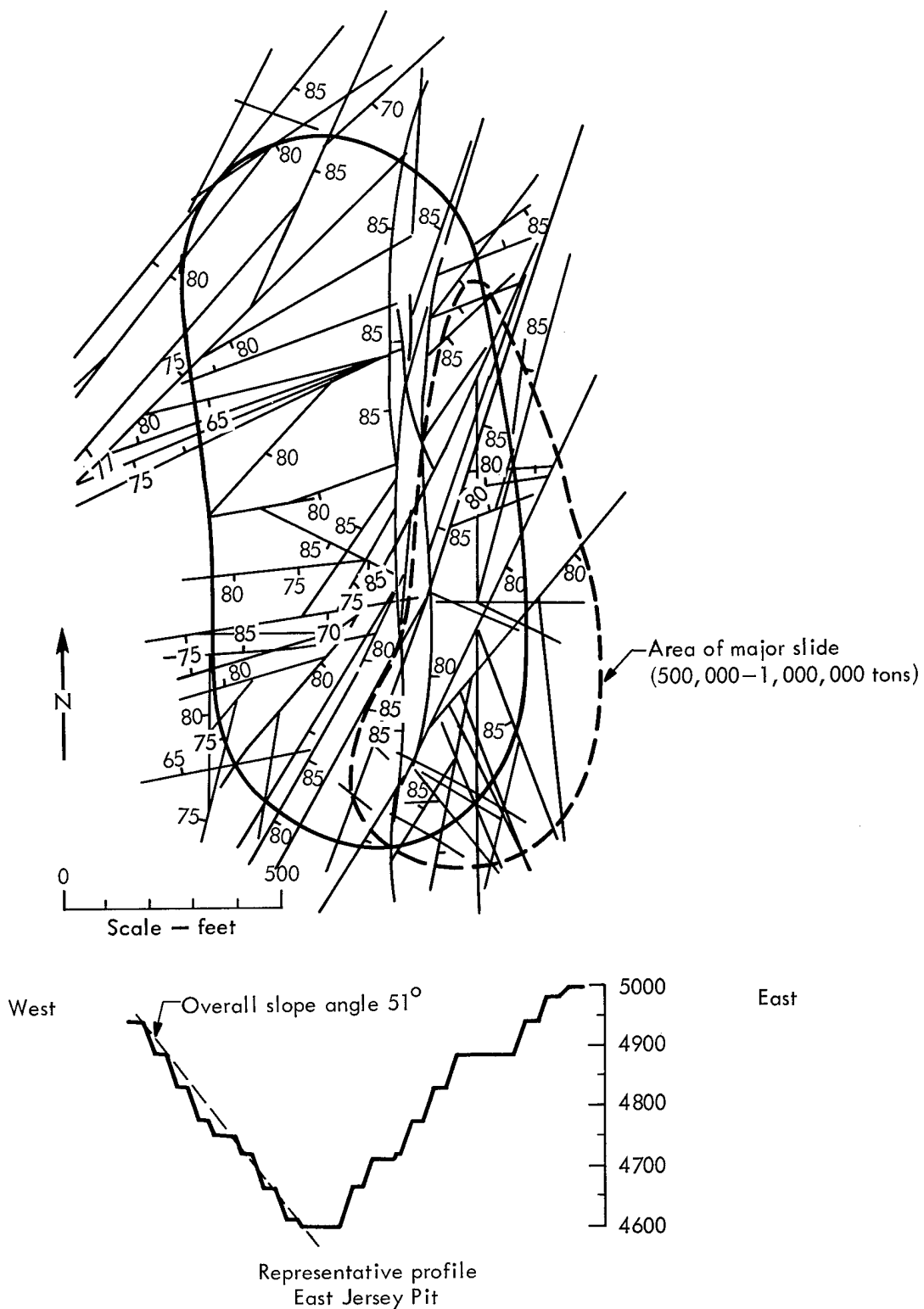


Figure A.27 East Jersey Pit; generalized plan showing fault orientations and slide area, and representative profile. (Adapted from data contributed by Bethlehem Copper Corporation, Ltd.)

"... to date there has been no occurrence of major slides..." in the Jersey Pit. Design of the Jersey pit calls for the development of a near-circular excavation averaging about 3,000 to 3,300 feet in diameter. Ultimate depth is expected to be about 1,000 feet. Pit walls are to be benched at average vertical intervals of about 33 feet, with individual bench faces sloped at about 70° and average overall pit wall inclinations of up to 45°.

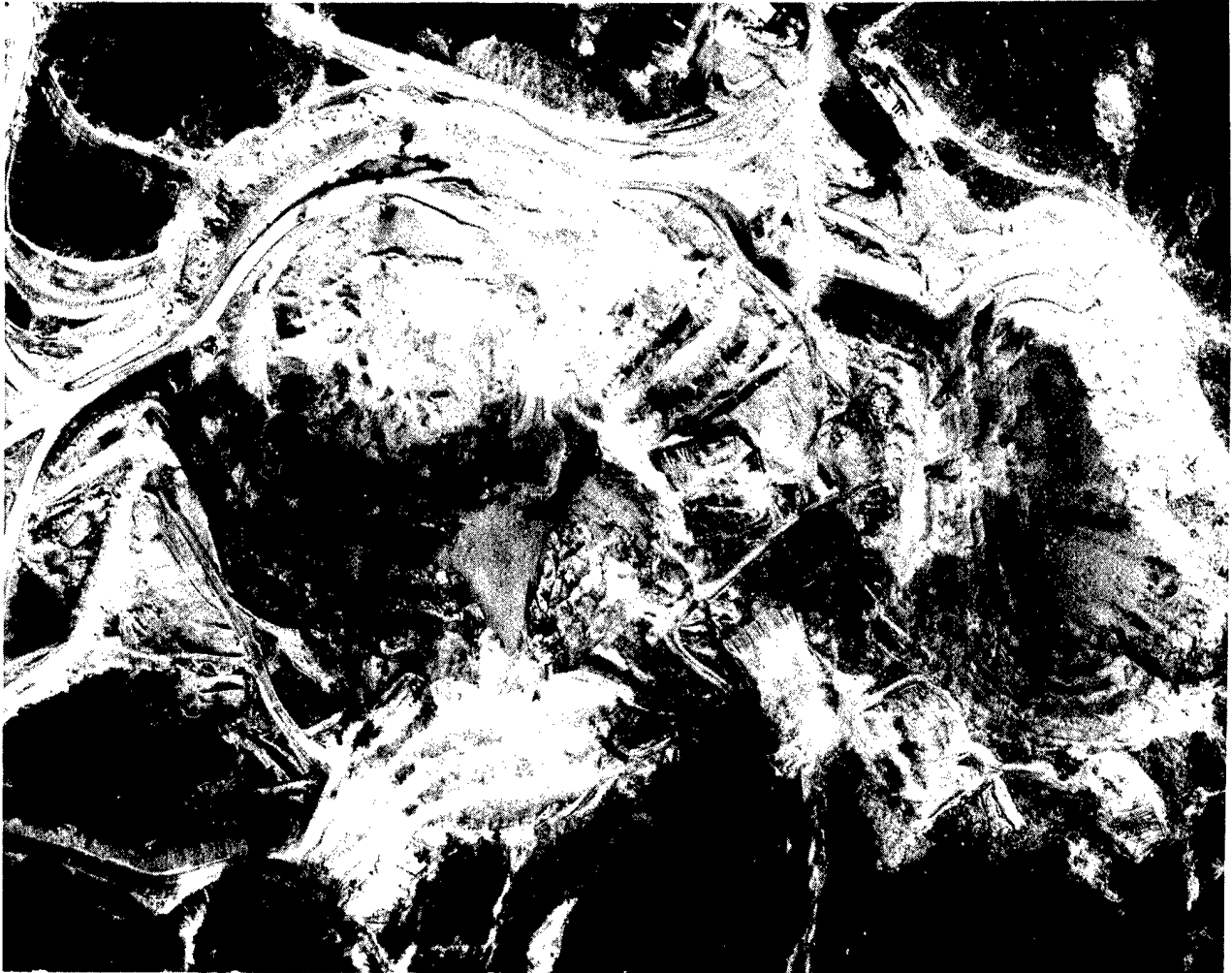


Figure A.28 East Jersey and Jersey Pits; vertical aerial view.
(Contributed by Bethlehem Copper Corporation, Ltd.)

ESPERANZA MINE
PIMA COUNTY, ARIZONA (22)

This open pit copper mine (Figure A.29) has been excavated to an approximate depth of 420 feet in porphyritic andesite, quartz monzonite, and quartz latite bodies which have intruded older pyroclastic rocks (Figures A.30 and A.31). Pit walls are



Figure A.29. Esperanza Mine; oblique aerial view, looking northward.
(Contributed by Duval Corporation.)

broken at vertical intervals of about 35 to 40 feet by benches which vary in width up to several hundred feet. The present average overall pit wall declivity is about 30° . Pit design calls for a final maximum depth of about 550 feet, with an average overall declivity of about 40° .

The area is cut by numerous faults, most of which follow a dominant NE-SW or a secondary NW-SE trend. Well-defined joint systems striking E-W and dipping north at 45 to 50° , striking N-S and dipping west at 60° , and striking NW-SE and dipping north-east at 55° are exposed in the eastern portions of the mine. These joint systems, however, are almost totally obscured in the western portions of the pit by a prominent NNW-SSE trending shear zone.

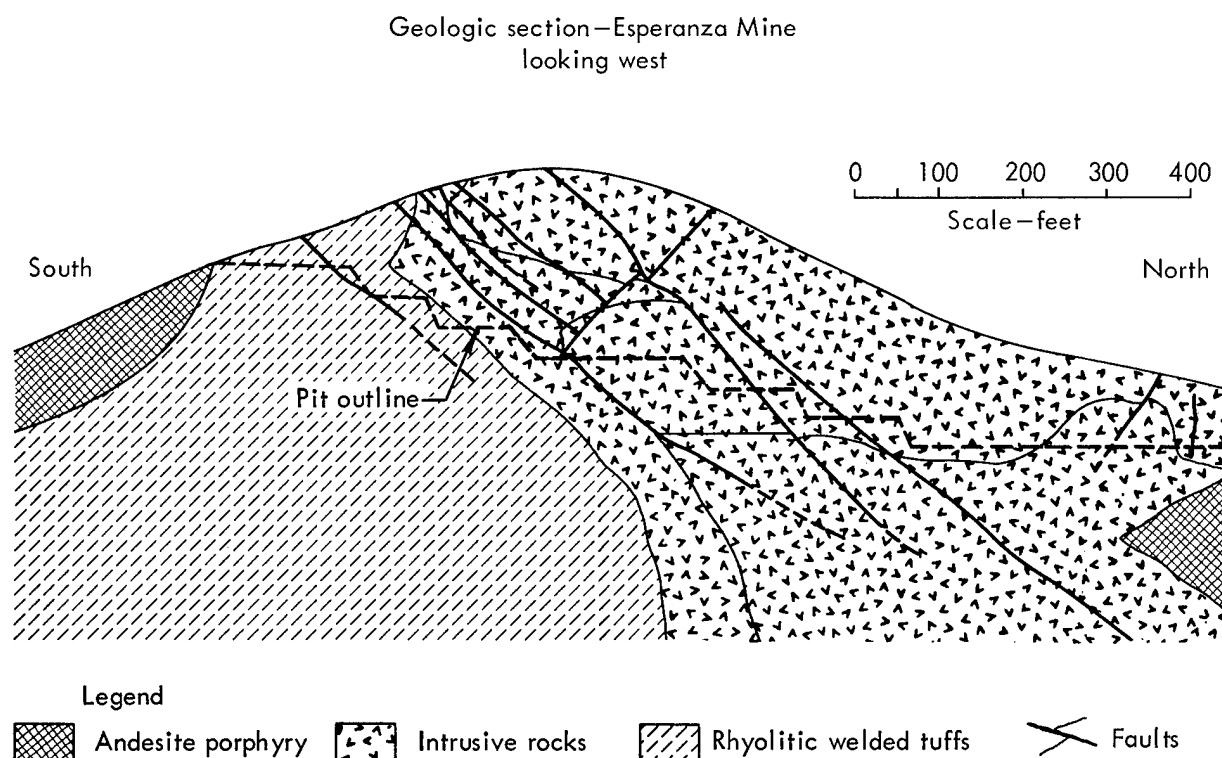


Figure A.30 Esperanza Mine; generalized geological section.
(Adapted from data contributed by the Duval Corporation)

Areas of sulfide mineralization are associated with zones of intense brecciation and argillic, sericitic, or siliceous alteration which may be significantly less competent than unaltered bedrock.

Some ground water flow into the pit is reported, and about 1 million gallons of ground water are pumped from the pit monthly.

The operating company has supplied the following description of stability conditions in the pits:

"There are two prominent subsidence conditions in the Esperanza Pit. The north slump is located in the northwest wall; occurs principally in material classified as quartz monzonite porphyry with a measured critical slope angle of 36 degrees; is coincident with inate structure, which defines the pit in this subsidence area; has been intensified by the removal of lateral support, resulting in a remnant spur; and, by the fact that the structure dips into the back slope, produces a characteristic dip and fault ('stairstep') disruption.

"The south slump, located in the southwest wall of the Esperanza pit is not believed to be tectonic in origin, although the bounding surfaces are defined by sheeting planes and a projection of the Cooper Fault, a major structure trending northeast, traverses the area. Occurring principally in material designated as rhyolitic welded tuff and quartz

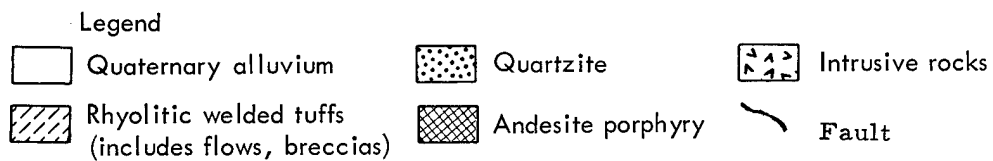
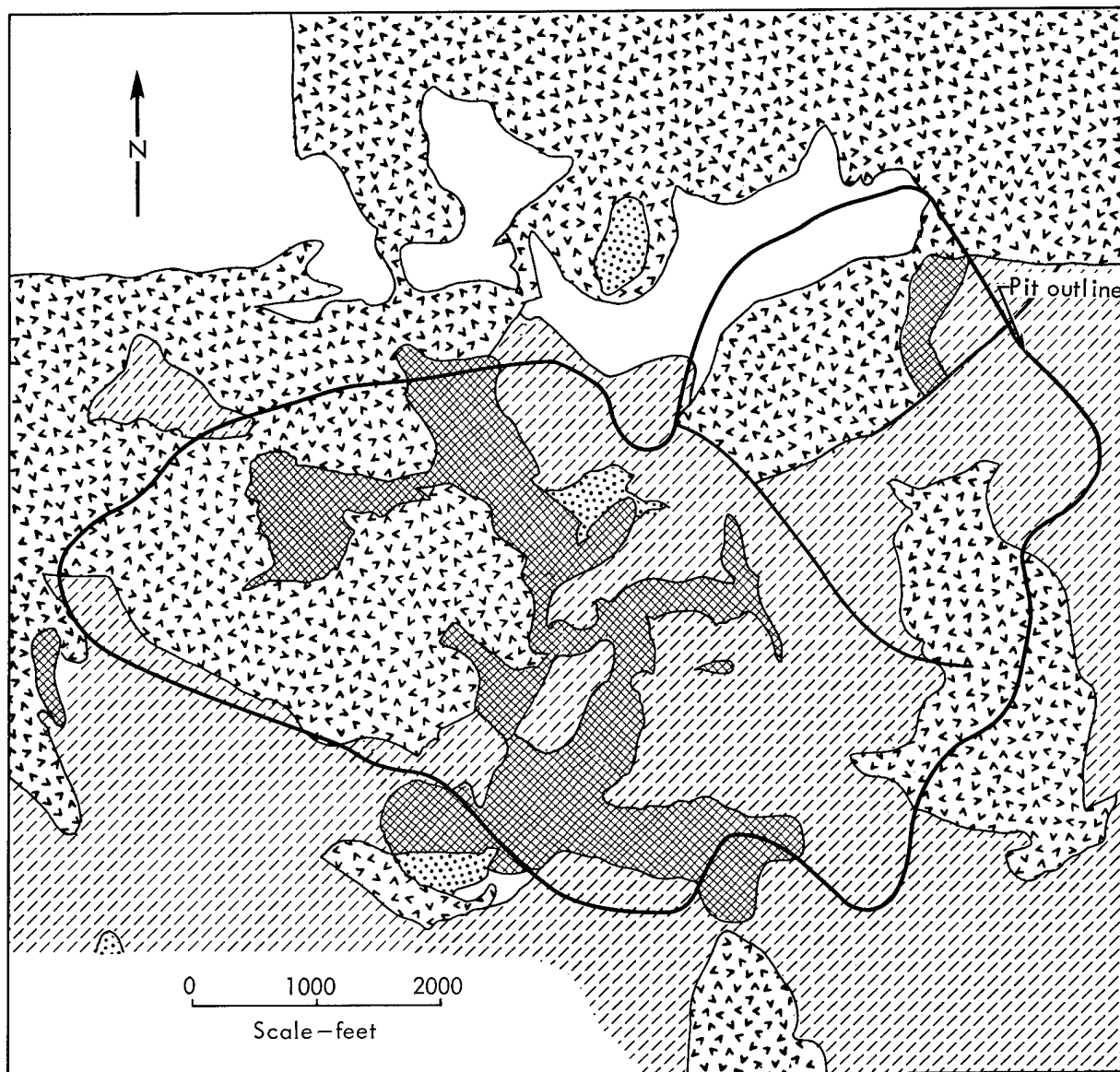


Figure A.31 Esperanza Mine; generalized geological map.
(Adapted from data contributed by the Duval Corporation)

latite porphyry (critical angle of 36 degrees), this subsidence, although partially translational, evidences rotational character. A prime cause is the removal of lateral support. Contributing causes are increased stress in relation to secondary permeability with limited cohesion of clay gouge material on bounding surfaces, excavation of toe support, lithostatic loading from capillary action, and back slope loading with fill. Failure acceleration reflects periods of seasonal heavy rainfall and excavation of toe material. This slump exhibits a graben and horst configuration, with a minor horst failing at a mean angle of 31.5 degrees and a major graben failing at 29.8 degrees.

"In the West Esperanza pit, wall failure occurs normal to the controlling and predominant northeast and east-northeast trending structure, i. e., where the back slope parallels structure, and poses no problem at present."

References 71, 114, 115, 171

FLETCHER QUARRY CHELMSFORD, MASSACHUSETTS (82)

This dimension stone quarry has been excavated to a maximum depth of 214 feet in granite. The bedrock shows prominent sheeting and jointing to depths of about 40 feet. Below this level, however, sheet thickness increases rapidly to more than 30 feet, and joint frequency diminishes markedly, so that the rock appears to be massive.

Working benches within the pit average about 50 feet in height. Individual bench faces are vertical or very nearly so, and the average overall slope is about 45° (Figures A.32 and A.33). No significant slope stability problems have been encountered in the 50-year history of this quarry.

The quarry is located in a region of gently rolling hills in which a formerly mature topography has been subdued by the abrasional and depositional effects of continental glaciation. The climate of the area is temperate, and the average annual precipitation is about 40 inches.

A problem of particular interest, from the standpoint of rock mechanics, is the relatively great amount of residual tectonic (compressive) stress present in the granite at this locality. This compressive stress has been calculated to be about 10,000 psi, and it is sufficient to produce a lateral expansion of about 1/2-inch per hundred feet

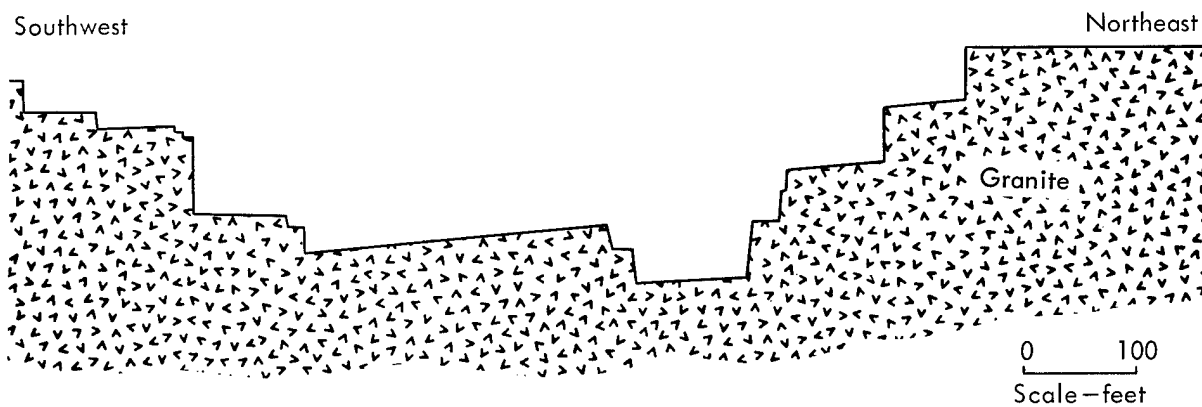


Figure A.32 Fletcher Quarry; representative profile.
(Adapted from data contributed by H. E. Fletcher Company)

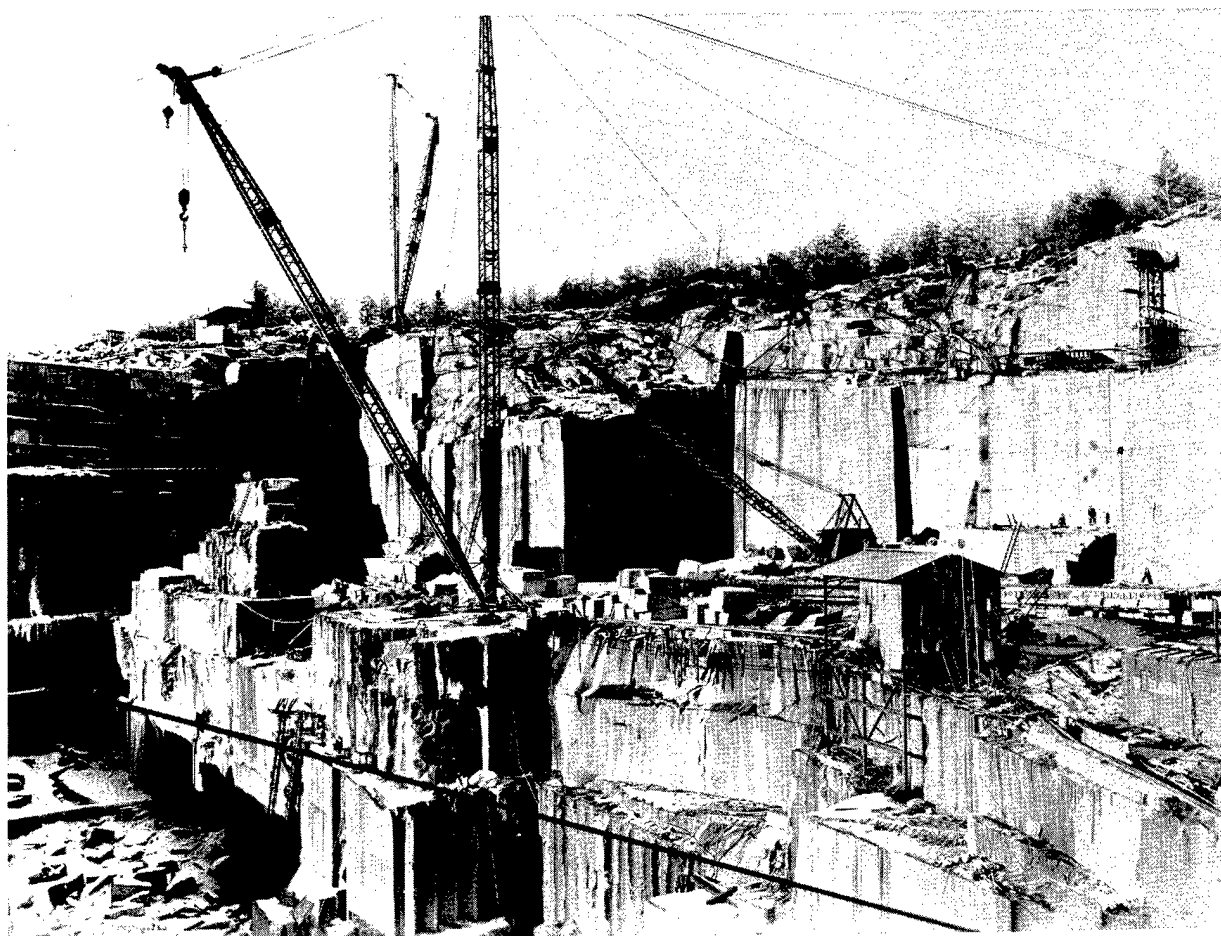


Figure A.33 Fletcher Quarry.
(Contributed by H. E. Fletcher Company)

when confining pressures are released in the course of quarrying operations. These residual stresses, especially when concentrated at sharp angular projections, are believed to be responsible for the occurrence of numerous rock bursts. These rock bursts, though of considerable significance from an economic and safety standpoint in the quarrying industry, are not of sufficient magnitude to be of significance in a consideration of overall slope stability.

It has also been noted in the course of operations at this quarry that the partial freeing of a large block from its matrix may result in a concentration of stresses within the intact cross section which exceeds the compressive strength of the rock and causes a crushing failure of the remaining connective web. The quarrying problems produced by this residual tectonic stress condition have been largely alleviated by the use of large-diameter calyx drills to excavate channels which isolate the quarry stone from the stress field of the surrounding bedrock.

References 9, 30, 31, 40, 174

KIMBLEY PIT RUTH, NEVADA (30)

The Ruth Mining District, also known as the Robinson or Ely Mining District, is situated in the Egan Mountains of east-central Nevada. Elevations in the area average about 7,000 feet, with peaks rising locally to about 8,000 feet. The climate is semiarid.

The area was organized as the Robinson District in 1868. Early mining activity involved small precious metal deposits. In 1900, an initial interest was developed in the low-grade "porphyry type" copper deposits of the district, and open pit operations were initiated in 1908. Four major open pit mines, the Veteran, Tripp, Liberty, and Kimbley pits, are located within the district.

Bedrock formations in the district include a thick sequence of sedimentary rocks (shales, sandstones, limestones, and dolomites), which are overlain by younger lavas and pyroclastics. The sedimentary sequence has also been invaded by intermediate to acidic intrusives. The various bedrock units of the district have been deformed by both folding and faulting. Ore-grade mineralization in the district is localized largely within an altered monzonite porphyry and adjacent sedimentary formations.

The Kimbley Pit is the eastermost of the four major open pit mines in this district. The pit has been developed in a mass of highly altered monzonite porphyry and metamorphosed shale which occurs along the footwalls of the Kimbley and

Jupiter faults (Figures A.34 and A.35). The pit is oval in plan, measuring about 1,500 by 1,200 feet, and the maximum pit depth is about 500 feet.

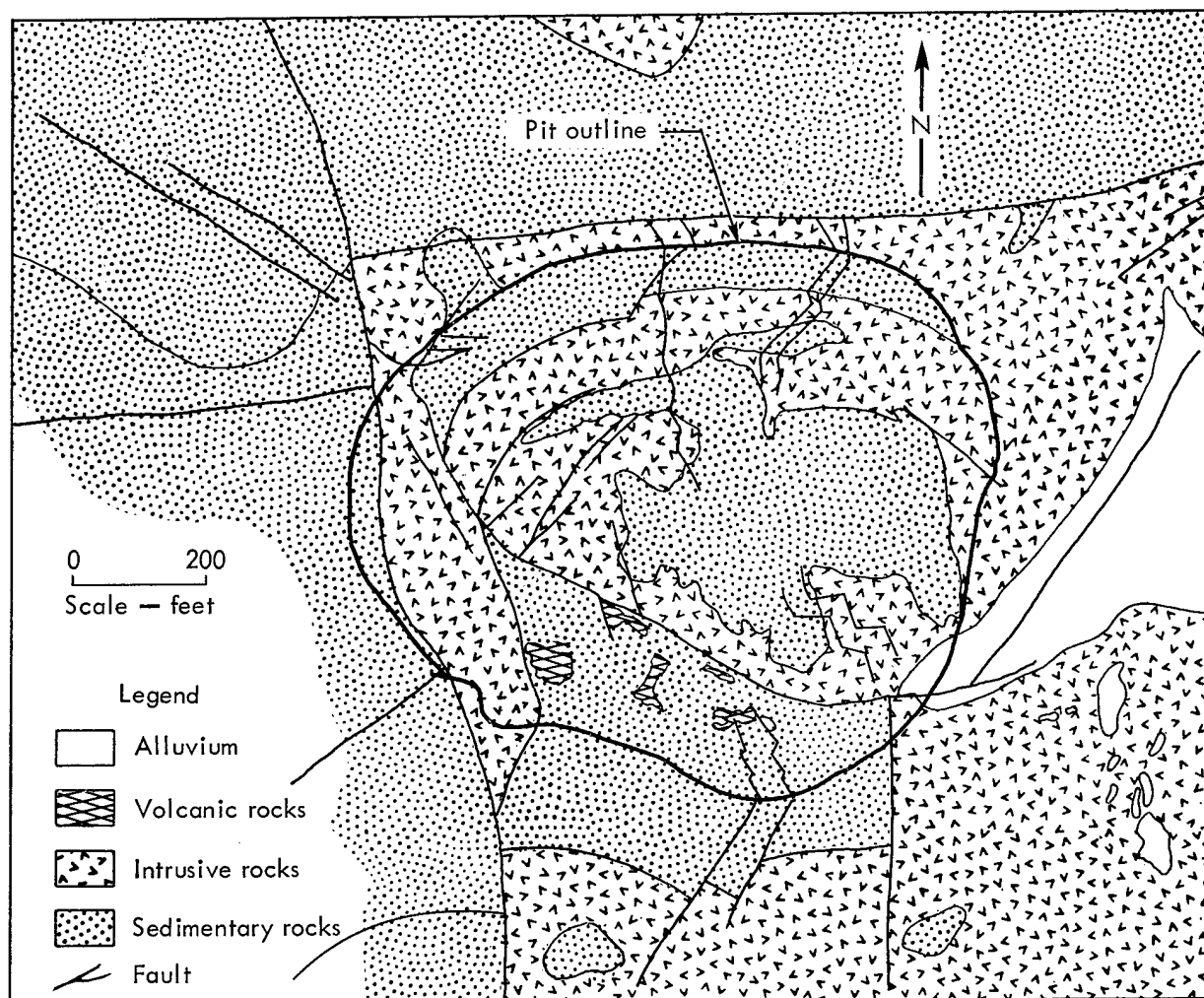


Figure A.34 Kimbley Pit; generalized geological map.
(Adapted from data contributed by Kennecott Copper Corporation)

In recent years, the Kimbley Pit has served as a full-scale test model in a joint Kennecott – U. S. Bureau of Mines slope stability study. This study involves a three-phase effort, as follows:

- a. Determination of structure and stress distribution in rocks adjacent to the pit, and employment of this data to determine maximum declivity of stable slopes
- b. Modification of existing slopes to calculated maximum declivity
- c. Oversteepening of slopes to induce failure.

The initial phase of this program, now completed, involved investigation of regional stress distributions using the overcoring stress relief method. Concurrently, a detailed structural analysis was made on the basis of surficial mapping, borehole instrumentation, and physical testing of oriented core samples. On the basis of this

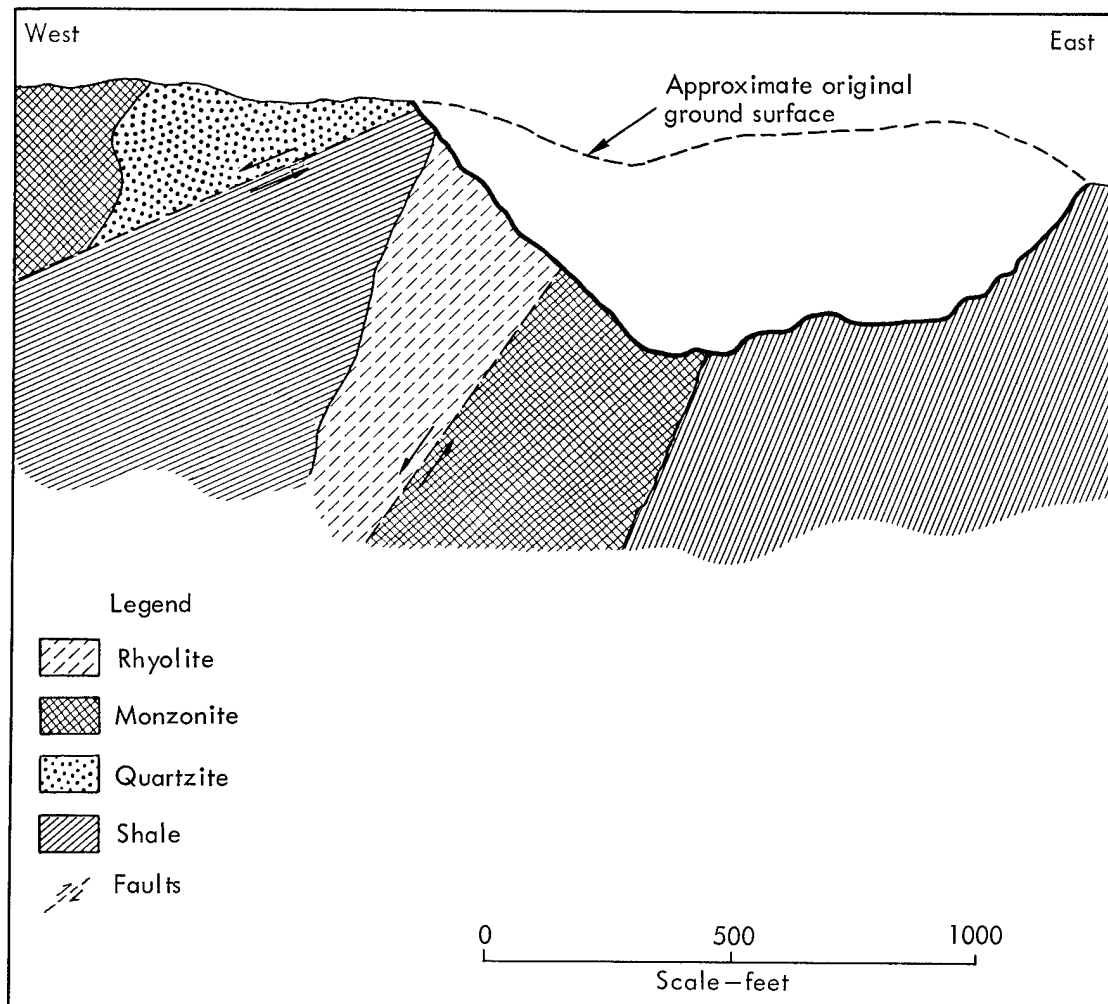


Figure A.35 Kimbley Pit; generalized geological section.
(Adapted by permission from Reference 134, Titley and Hicks, eds., 1966)

research, it was determined that there was little residual tectonic stress in the rocks adjacent to the pit, and it was found that the most prevalent structural anisotropy in the northern and western walls of the pit consisted of a joint set dipping toward the pit at an angle of approximately 60° .

The second phase of the program, completed in the fall of 1966, involved steepening of the western and northern walls of the pit (Figure A.36) from 45° (average overall declivity at termination of mining operations) to between 58 and 62° (approximately equal to the dip of the major joint set in rocks of the pit wall) (Figure A.37). Concurrently, two parallel adits driven into the northwestern pit wall were instrumented and monitored to detect any local stress variations which might indicate an impending slope failure. (This monitoring is still being carried on; however, there have been no significant slope failures during or since the steepening operation,

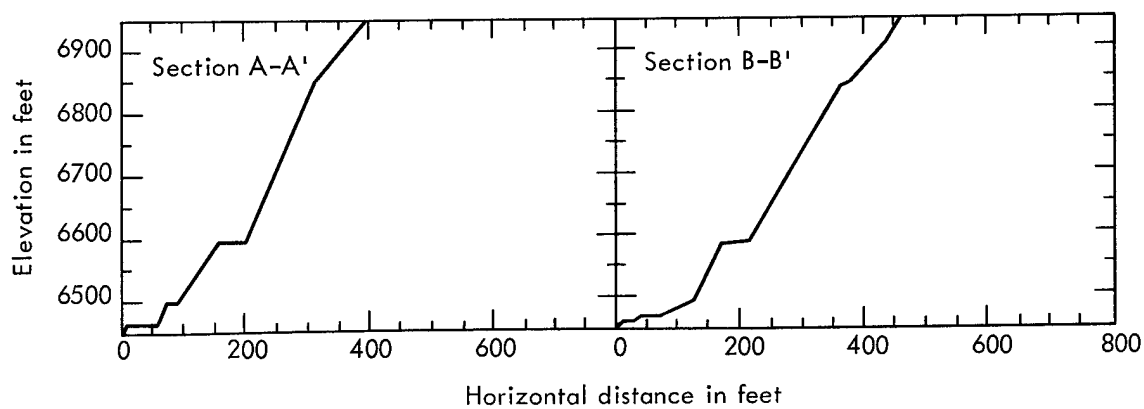
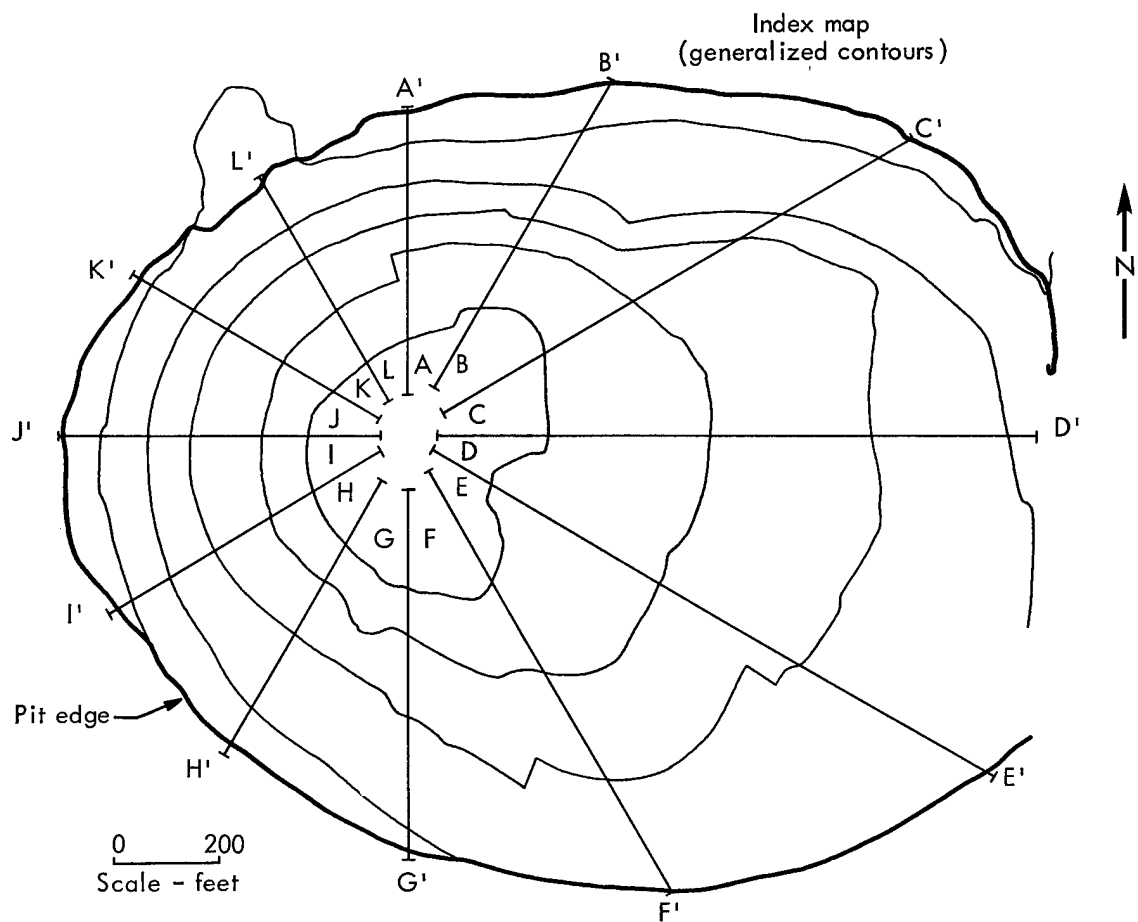
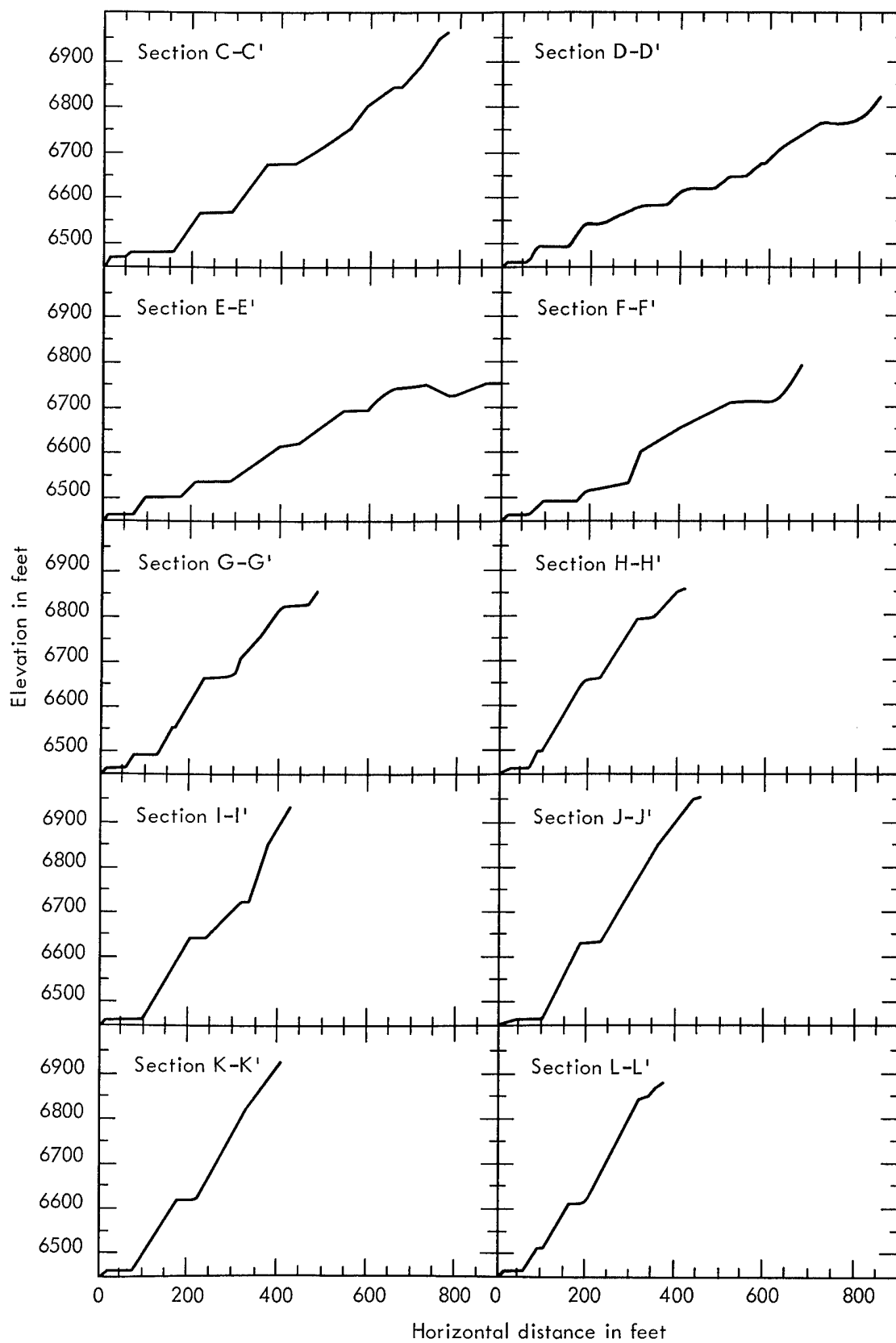


Figure A.36 Kimbley Pit; generalized plan and representative profiles.
(Adapted from data contributed by Kennecott Copper Corporation)



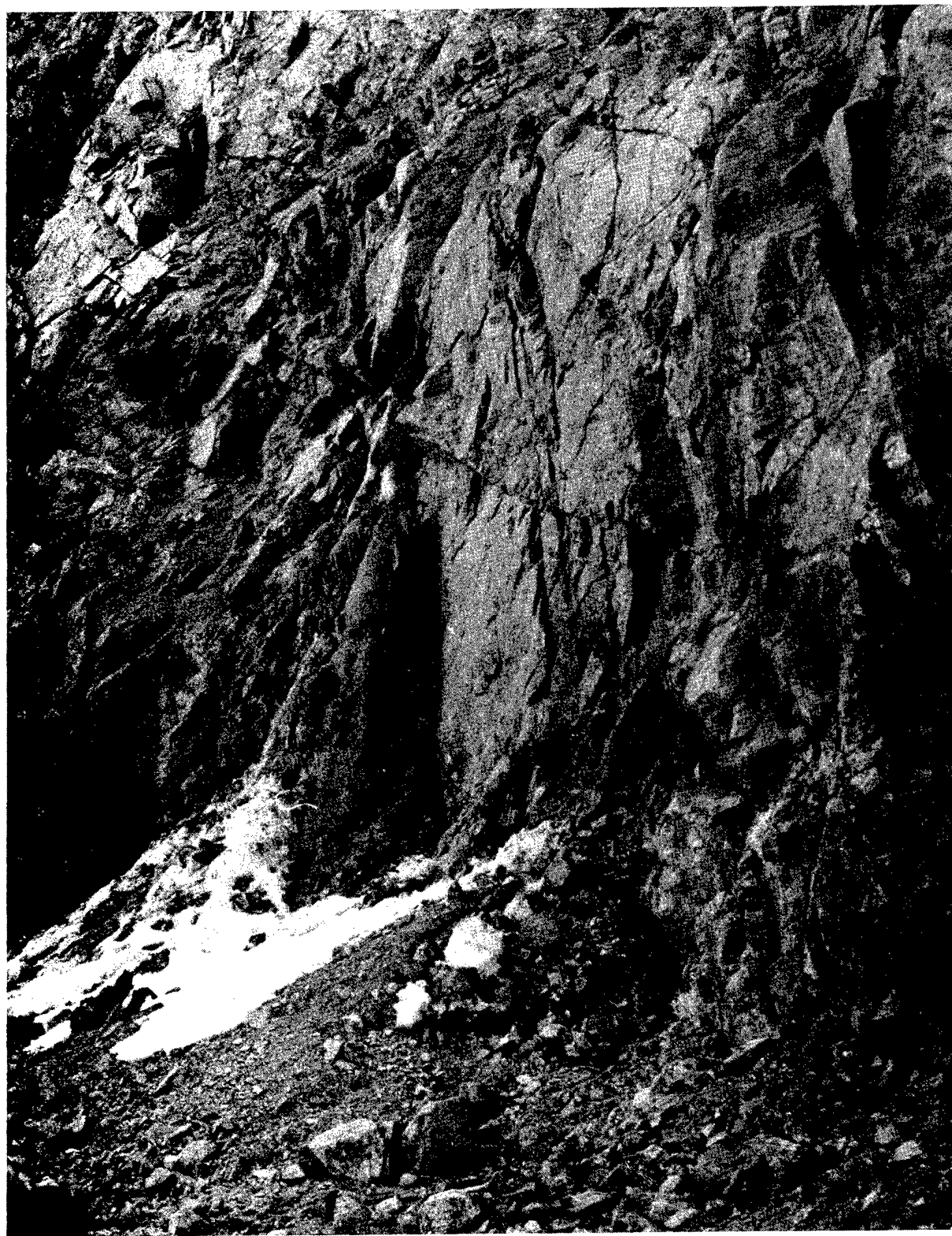


Figure A.37 Kimbley Pit; base of northwestern wall, showing bedrock jointing.
(Reproduced by special permission of Kennecott Copper Corporation)

though minor surficial sloughing of small rock fragments takes place almost constantly.) A surficial slide of minor magnitude took place in the southern wall of the pit upon completion of excavation, but this was localized in an extremely incompetent argillic rhyolite.

The third phase of the program, in which failure of the pit slopes will be deliberately induced and monitored, is still in the initial planning stages. It has not yet been decided where, when, or how this phase of the study will be implemented.

References 8, 10, 11, 56, 67, 86, 101, 102, 123, 124, 145, 179

KIRUNA PIT KIRUNA, SWEDEN (31)

The open pit iron mine at Kiruna, which measures approximately 2-1/2 miles in length and up to 1,000 feet in width (Figure A.38), has been excavated to depths of up to 750 feet. The relatively small portion of the pit which is still in operation is expected to reach a maximum depth of about 400 feet.

The ore consists of a 300 foot wide sheet-like body of massive magnetite which strikes NNE-SSW and dips eastward at about 60°. Massive porphyritic rocks form both the hanging wall and the footwall of the deposit.

Pit walls are broken at vertical intervals of up to 100 feet by benches up to 20 feet in width. Individual bench faces are sloped at 68°, and average overall pit wall declivity varies from 45 to 55° (Figures A.39 and A.40).

The major structural features of the pit walls are two systems of joints and faults. One of these (dominant) strikes NW-SE and dips moderately to steeply toward the southwest. The other (secondary) strikes NE-SW and dips moderately toward the northwest or southeast. Joints and faults of both systems range in thickness up to 40 inches, and are usually filled with mineral matter. Occasionally they are open and water bearing. Two major brecciated fault zones, which vary in width from about 3 to 16 feet, strike WNW-ESE across the ore body and dip steeply southward.

The climate at Kiruna is subarctic and subhumid. The local water table, represented by the level of nearby lakes, is approximately equal in elevation to the present pit floor. Substantial ground water seepage into the pit is reported, especially in the northwestern portion of the excavation which lies in close proximity to Lake Luossajärvi (Figure A.41).

In general, the stability of the pit walls is described as "extraordinary (sic) good," and no serious slope stability problems have been encountered during the life of the

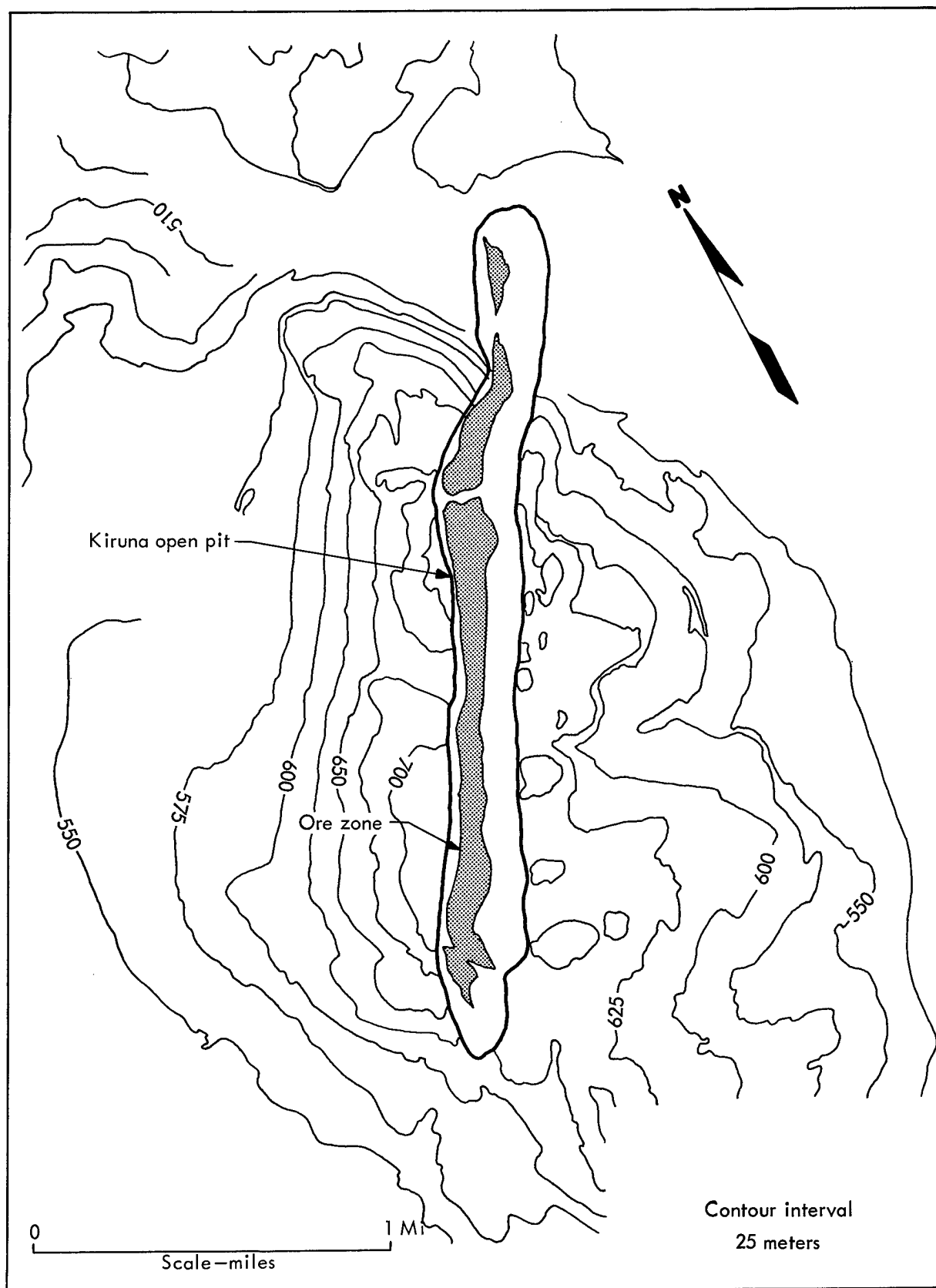


Figure A.38 Topographic Map; vicinity of Kiruna Pit.
(Adapted from data contributed by Luossavaara-Kiirunavaara Aktiebolag)



Figure A.39 Kiruna Pit; view northward from southern end, 1957.
(Contributed by Luossavaara-Kiirunavaara Aktiebolag)

mine (since 1900) (Figure A.42). The minor stability problems which are occasionally encountered are attributed to the presence of argillaceous and/or micaceous materials along joint or fault planes.

In recent years open pit operations at Kiruna have been largely phased out in favor of underground sublevel caving at greater depths. In those portions of the pit where open-cut operations have been suspended and underground operations are being

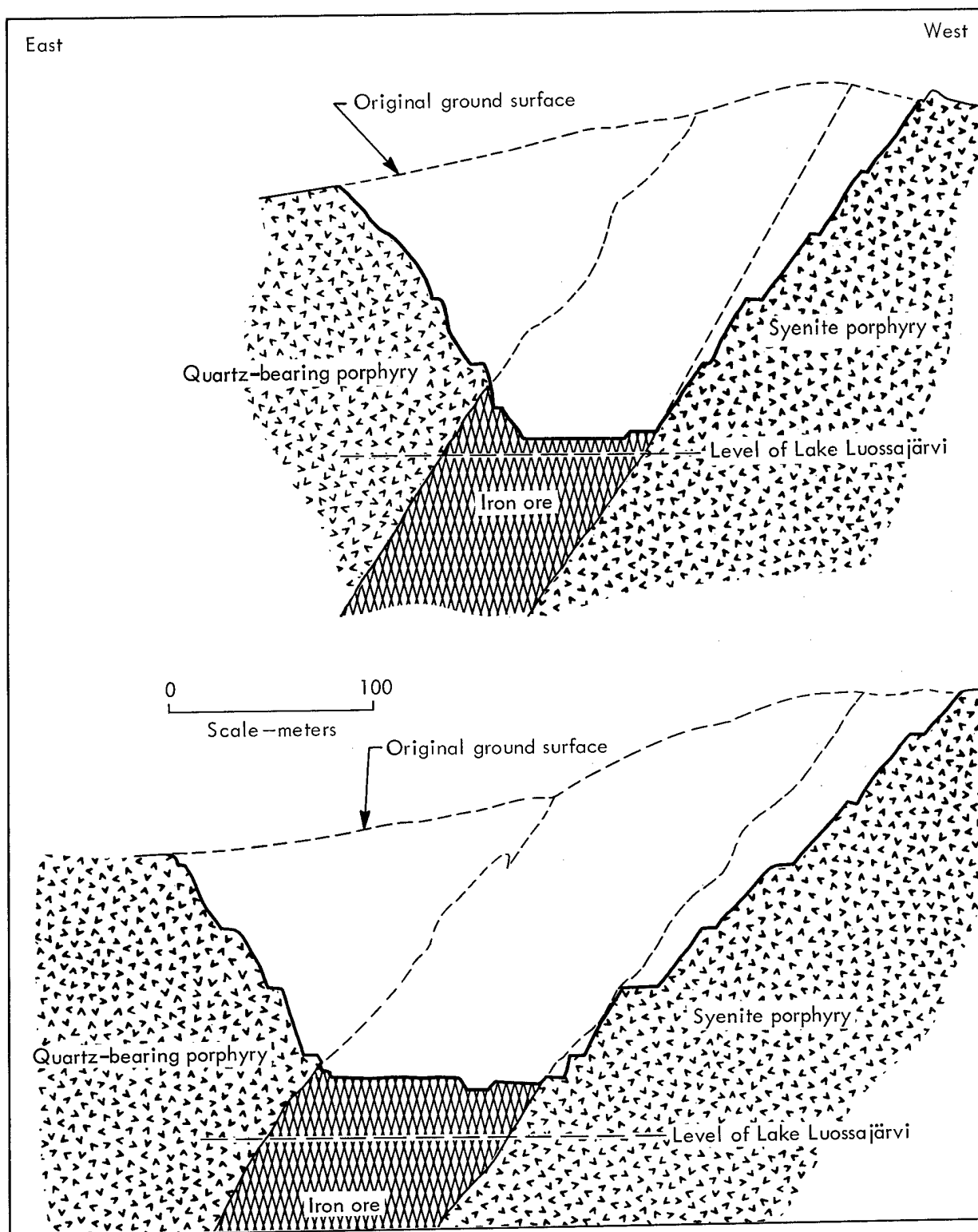


Figure A.40 Kiruna Pit; representative profiles and geological sections.
(Adapted from data contributed by Luossavaara-Kiirunavaara Aktiebolag)



Figure A.41 Kiruna Pit; view from the north, 1959.
(Contributed by Luossavaara-Kiirunavaara Aktiebolag)

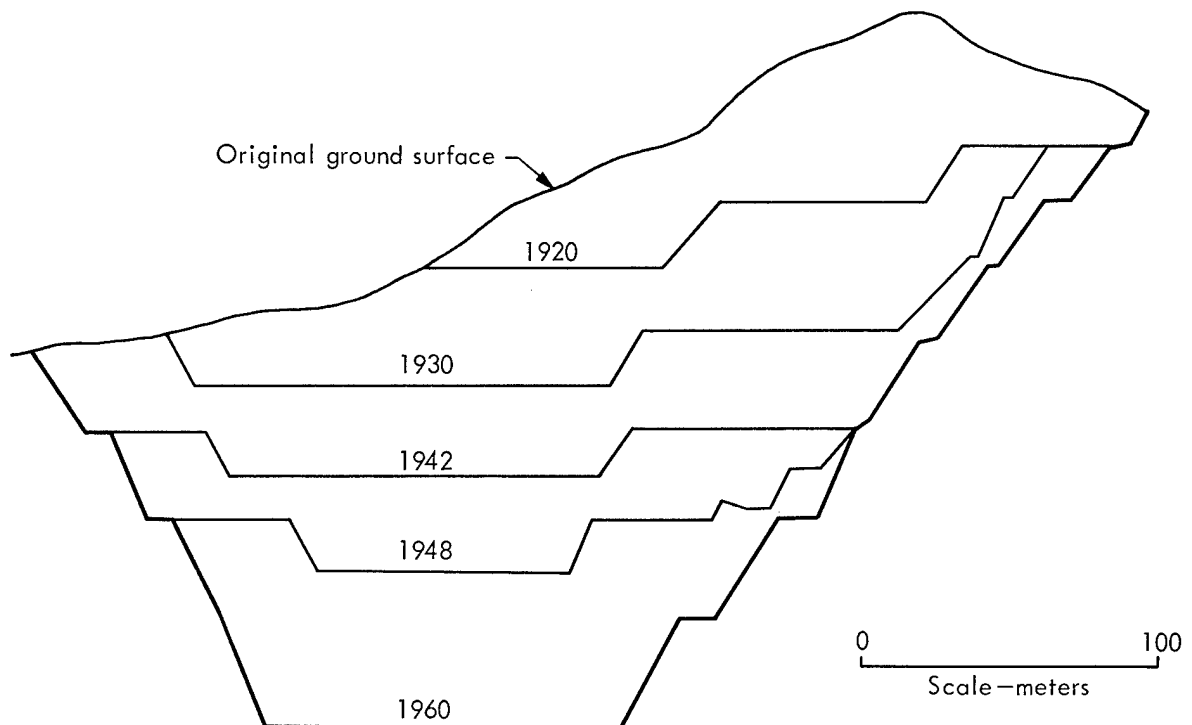


Figure A.42 Kiruna Pit; representative profile, showing progress of excavation from 1920 to 1960. (Adapted from data contributed by Luossavaara-Kiirunavaara Aktiebolag)

carried on, the configuration of the hanging wall of the pit has been substantially altered. Broken debris caved from the hanging wall stands at an angle of repose of about 30 to 40°; the boundary between rubble and intact rock is believed to slope at 80 to 90°; and the plane which separates the zone of subsidence-induced jointing from that of natural jointing slopes at about 70° (Figure A.43).

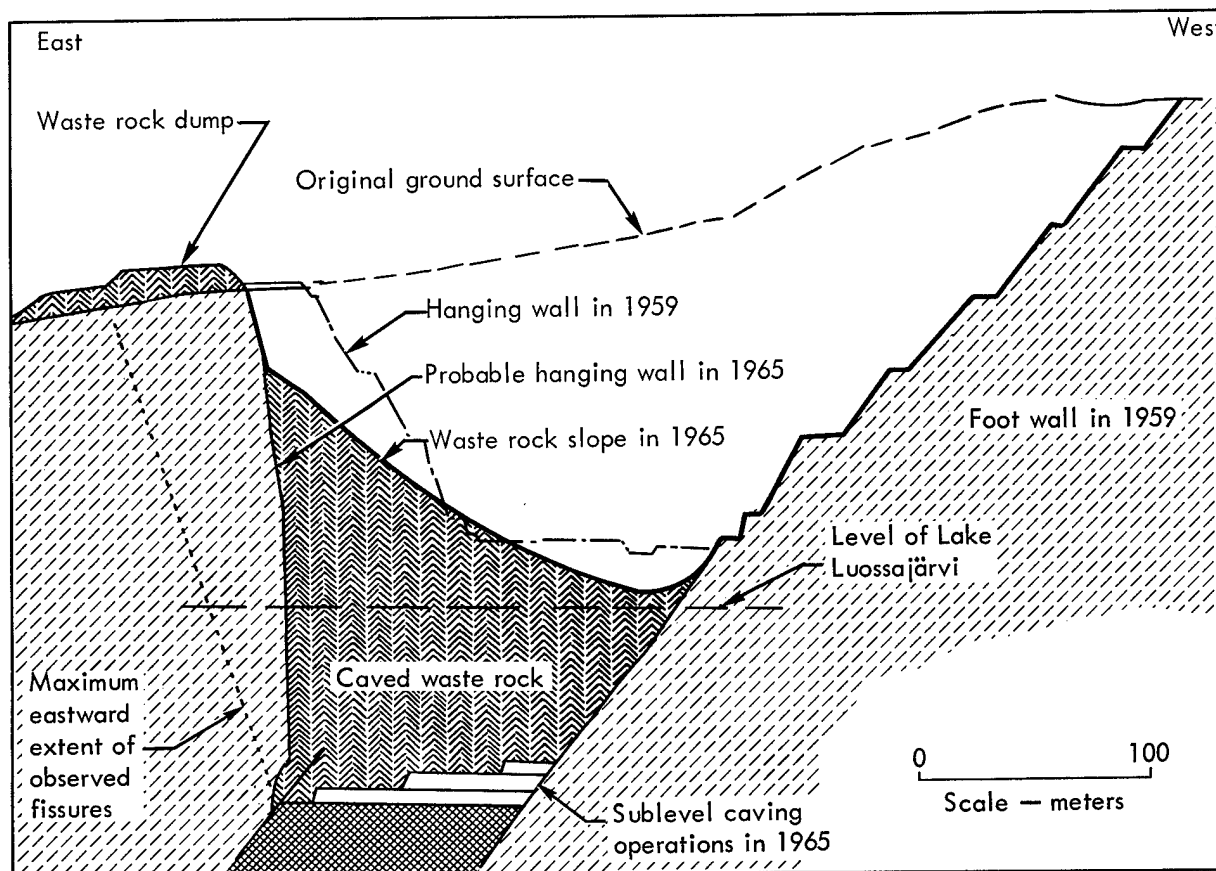


Figure A.43 Kiruna Pit; representative profile and section showing modification of pit walls by collapse due to sublevel caving. (Adapted from data contributed by Luossavaara-Kiirunavaara Aktiebolag)

References 8, 21, 38, 67, 180 (cf. Figure 1.2)

LIVE OAK AND THORNTON PITS GLOBE, ARIZONA (36, 67)

These open-pit copper mines are located 6 miles west of Globe, Arizona, in an area of rugged topography at the southern extremity of the Mazatzal Mountains. The climate of the region is semiarid, with an average annual precipitation of 20.3 inches, and an annual temperature range of 13 to 110°F. Most of the rainfall occurs during thunderstorms in July and August.

The first discovery of surface mineralization in this district was made in 1872, and the first copper prospects were worked in 1881. Two open pit mines, the Live Oak and Thornton Pits, are currently in operation. Subsurface mining of the Live Oak and Thornton ore bodies preceded the current open pit operations, and block-caving activity was continued until 1954.

The Live Oak Pit measures approximately 3,000 by 1,200 feet in plan and is about 600 feet deep. The Thornton Pit measures about 2,500 by 2,500 feet in plan and has a maximum depth of about 1,050 feet (Figure A.44). Operating benches in both pits are

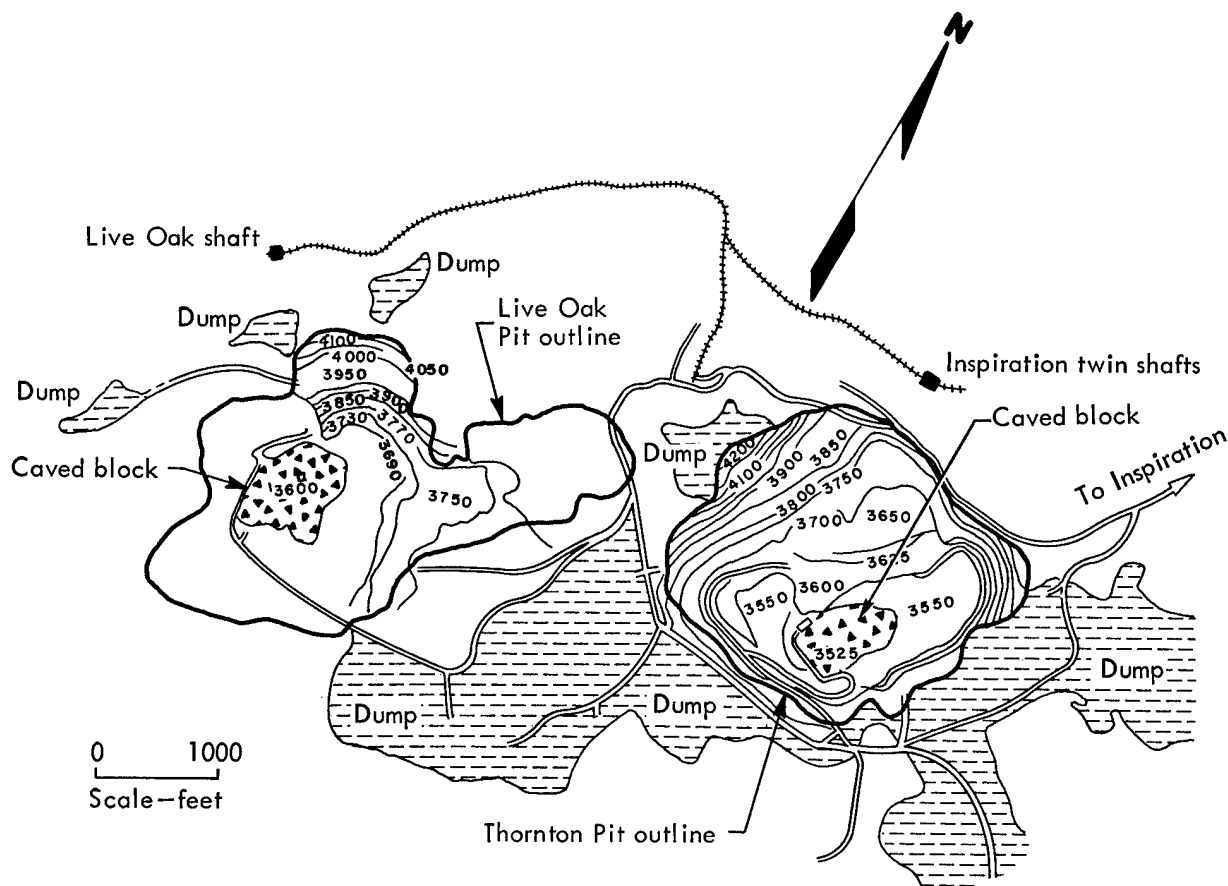


Figure A.44 Live Oak and Thornton Pits; generalized plan showing location of block-caved zones and dump material. (Adapted from Reference 49, Hardwick, 1963)

about 50 feet high and 25 feet wide. The average overall slope of pit walls is about 45° , but this declivity has been reduced to about 37° in areas of actual or potential slope instability (Figures A.45 and A.46).

Bedrock lithology of each pit includes schists which have been transected by several plutonic and volcanic bodies of intermediate composition. Ore grade mineralization is localized in a porphyritic granite and in adjacent portions of the schist.

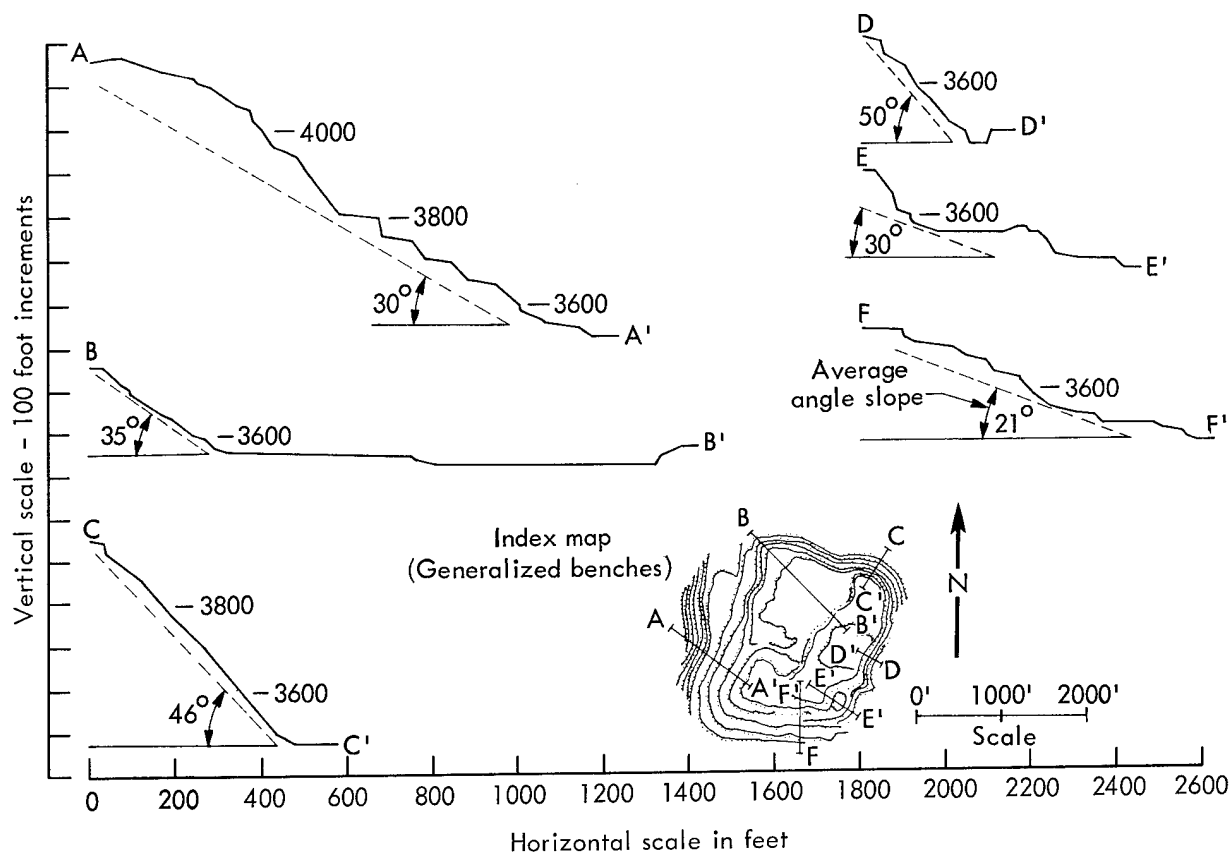


Figure A.45 Thornton Pit; generalized plan and representative profiles.
(Adapted from Reference 49, Hardwick, 1963)

Several major faults are found within or adjacent to the pits. The Bulldog fault on the northwestern side of the Thornton Pit slopes toward the excavation at about 35° , and cracks have appeared at the rim of the pit in this area (Figure A.47).

A major slide area exists in the northwestern portion of the Live Oak Pit (Figure A.48). Movements here were first noted in 1955 and have continued to the present time. This area involves both undisturbed bedrock and rubble zones resulting from earlier block-caving operations. Heaving of the slide base is particularly prominent at the foot of slopes in undisturbed bedrock. Mine personnel have suggested that the zones of block-caving rubble may be more stable than the undisturbed bedrock because adversely oriented planes of weakness in the in situ rock have been destroyed or

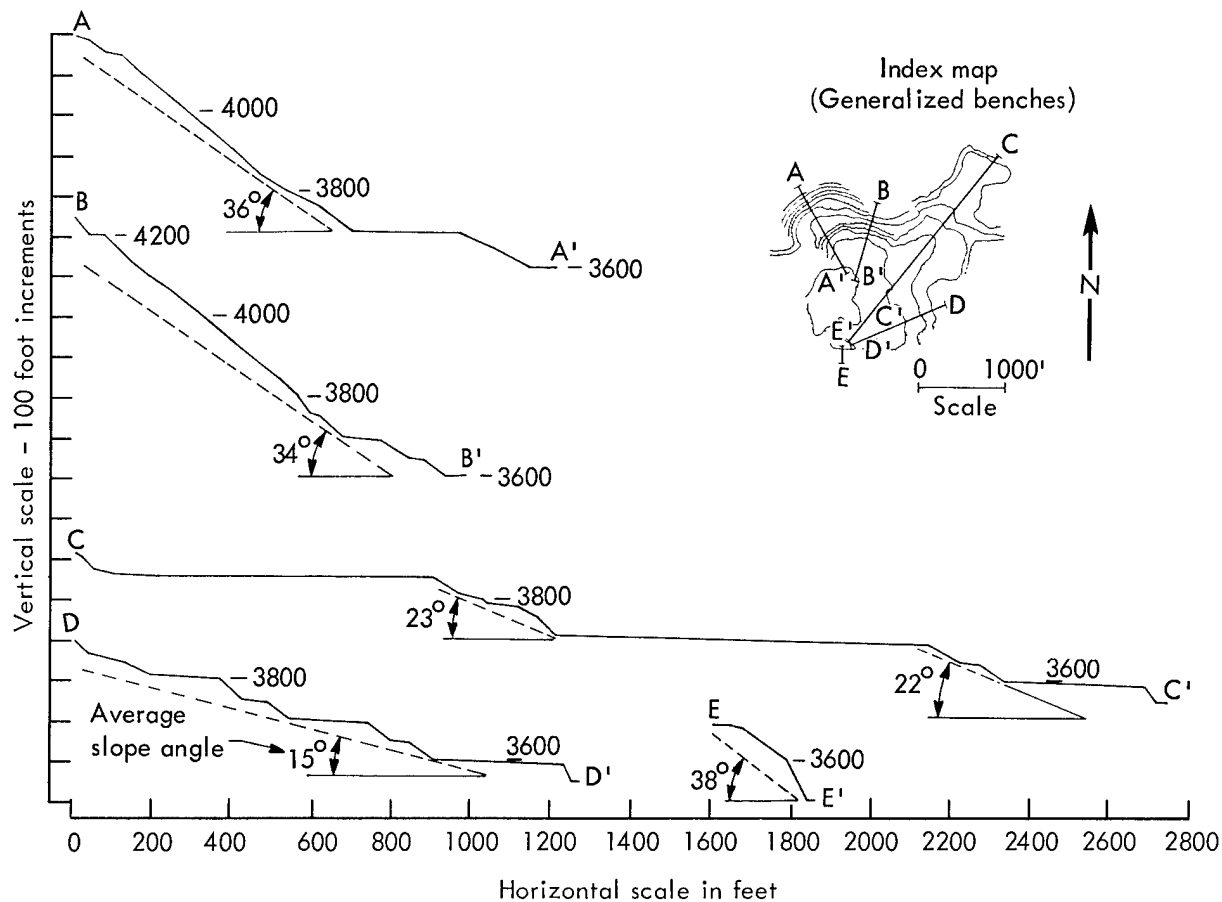


Figure A.46 Live Oak Pit; generalized plan and representative profiles.
(Adapted from Reference 49, Hardwick, 1963)

masked by diversely oriented fractures induced by the block-caving operations. Adversely oriented fault planes are thought to have been the prime cause of slope instability in these pits. The effects of ground water, though difficult to assess quantitatively, are also thought to have been a significant contributing factor.

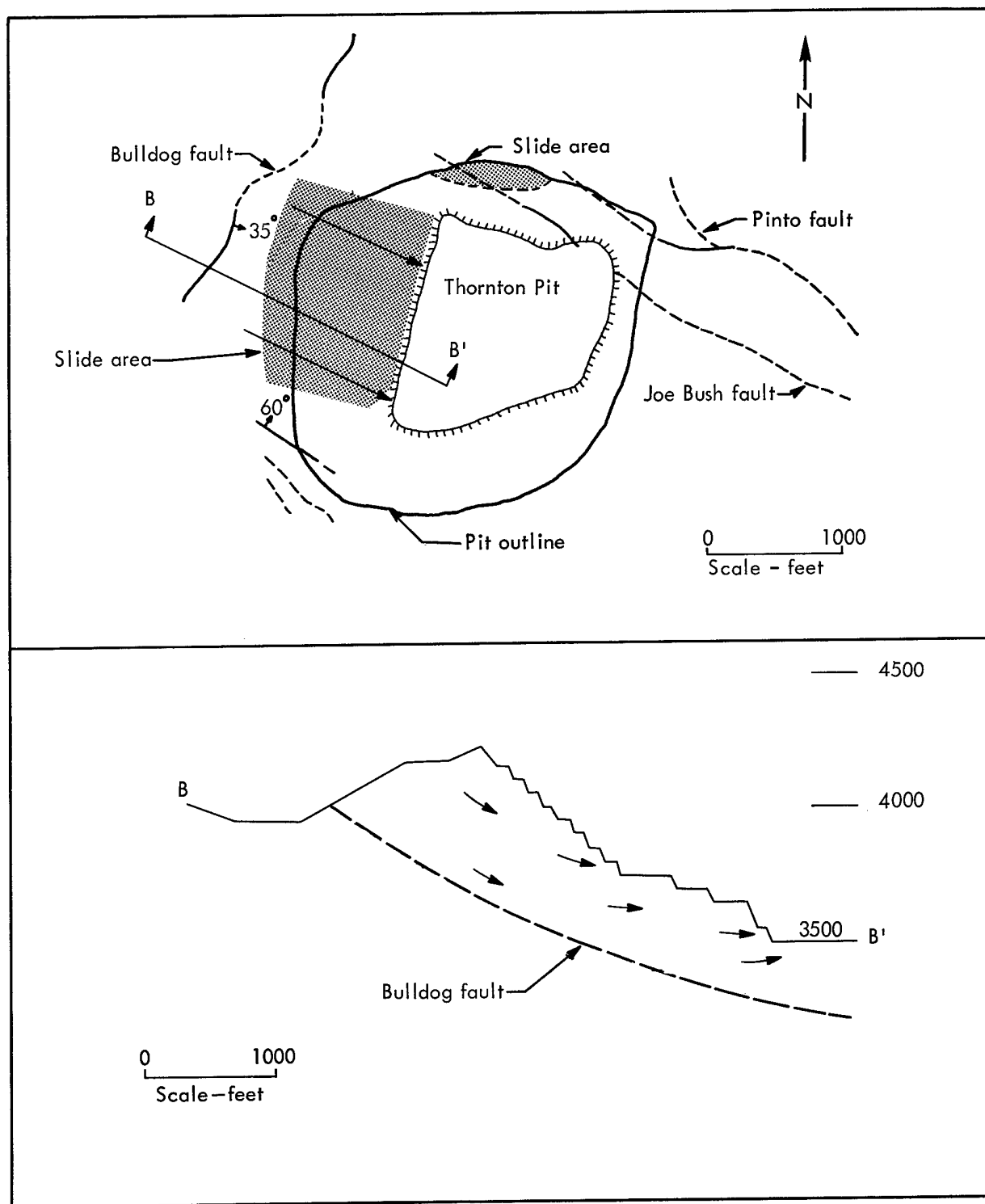


Figure A.47 Thornton Pit; generalized plan showing slide area, and generalized section through slide. (Adapted from data contributed by Inspiration Consolidated Copper Company)

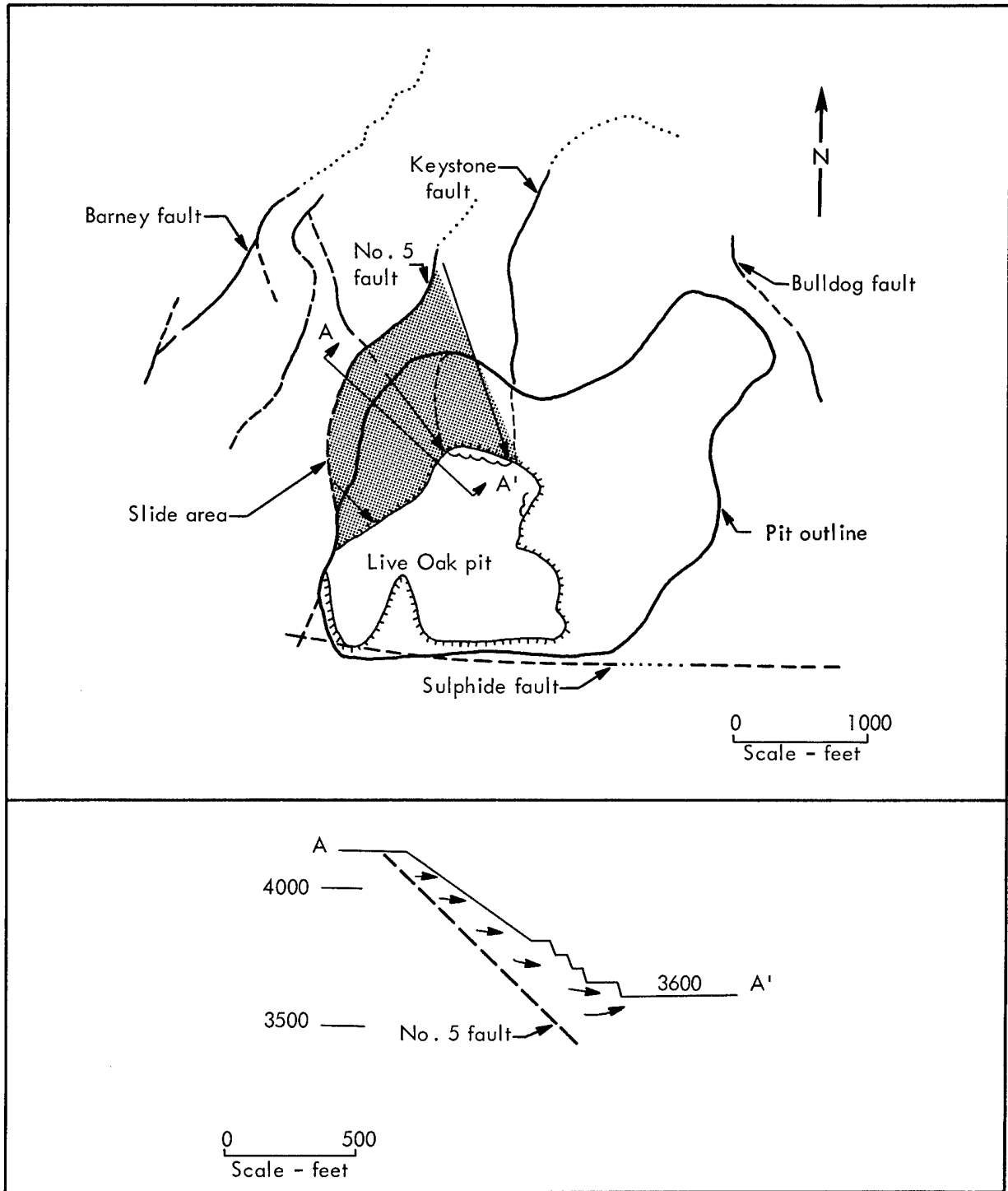


Figure A.48 Live Oak Pit; generalized plan showing slide area, and generalized section through slide. (Adapted from data contributed by Inspiration Consolidated Copper Company)

MISSABE MOUNTAIN MINE
VIRGINIA, MINNESOTA (40)

This open pit iron mine, measuring about 1,200 feet in length and up to 400 feet in width, has been excavated to depths of up to 450 feet in horizontally bedded cherty and slaty metasedimentary "iron formations" and their residual weathering products.

The mine has been discontinuously operated since the 1920's. By 1945 the northeast wall of the pit had reached a height of 350 feet and had been cut to an average slope of 65° throughout its lower portion. (A benched slope, about 50 feet in height and with an average angle of 25°, was established in glacial overburden and near-surface bedrock.)

Between 1945 and 1947 the depth of excavation was increased by 100 feet, and this lowest portion of the northeast wall was sloped at about 75 to 80°.

During a period of inactivity which extended from 1947 to 1960 there were no significant slope adjustments, though some surficial sloughing produced a talus accumulation which stood at 30 to 35° to a height of 50 to 75 feet at the toe of the slope.

Relatively small-scale scavenging operations which were carried on between 1960 and 1962 resulted in a slight lowering of the pit floor and a cutting back of the northeastern pit wall. In the course of these operations the average overall slope of the unbenched northeast wall was increased slightly (to about 70°) without any stability problems (Figures A.49 and A.50).



Figure A.49 Missabe Mountain Mine; view of northeastern wall, showing location of Figure A.50 sections. (Contributed by Pittsburgh Pacific Company)

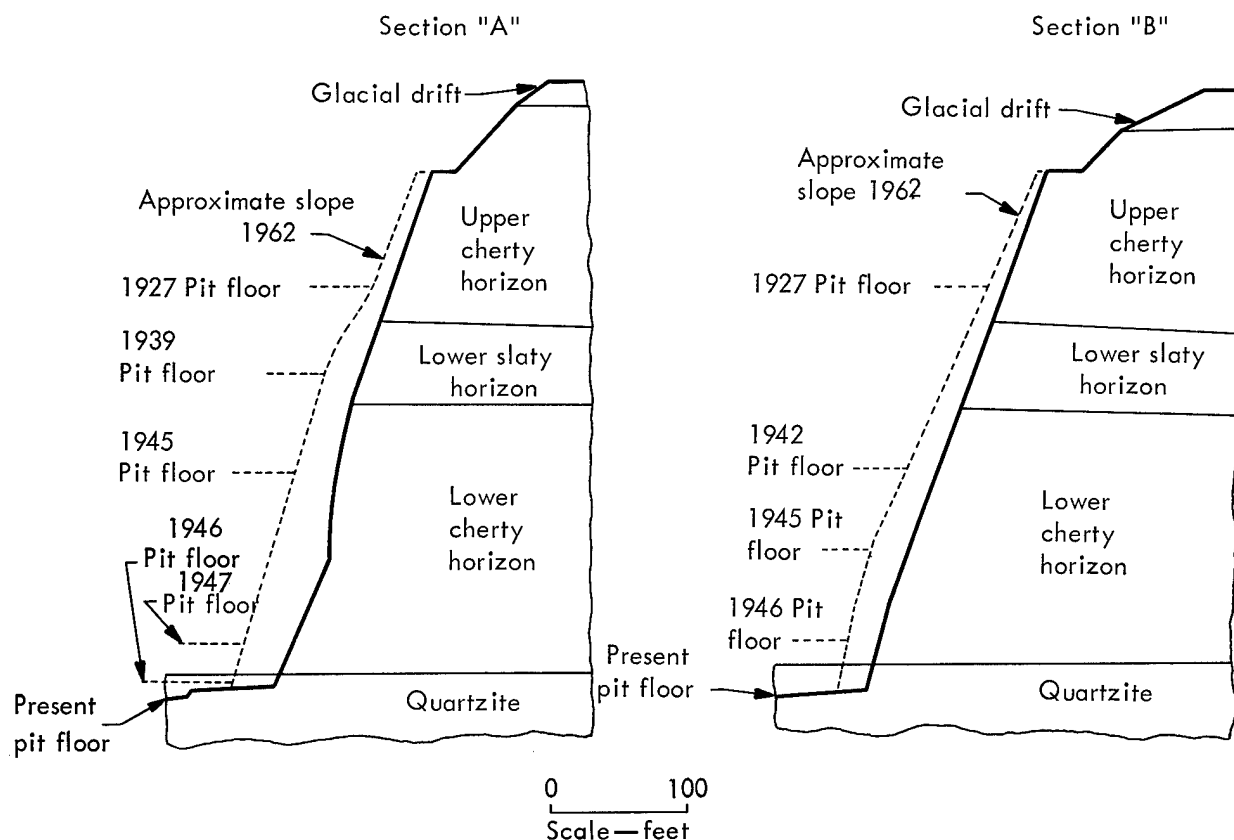


Figure A.50 Missabe Mountain Mine; representative sections.
(Adapted from data contributed by Pittsburgh Pacific Company)

MULHOLLAND CUT LOS ANGELES, CALIFORNIA (116)

This 3,200-foot-long cut, along a portion of the San Diego Freeway in Los Angeles, reaches a maximum depth of 260 feet in interbedded shales and poorly consolidated sandstones. These sedimentary strata strike generally E-W (approximately at right angles to the longitudinal axis of the cut) and dip northward at moderate angles.

The excavation site lies between 1,000 and 1,500 feet above sea level in an area of low but rugged relief. The climate is semiarid and subtropical. The water table, prior to excavation, was located about 120 feet above the proposed grade.

Initial design, based upon empirical consideration (45° slopes had remained stable in a large nearby borrow pit), called for an average slope angle of 35°. Slopes were to be broken by 30-foot berms at 60-foot vertical intervals, with individual benches being sloped at 45°.

Excavation, based upon this design, was started at the northern end of the cut; the intention being to progress southward through the hill while keeping the floor of the excavation at or near grade. Several minor slides developed during the initial stages

of excavation, and slope stability problems became more numerous and also more serious as work progressed. Finally, with about one-third of the cut completed, a massive failure occurred with a portion of the western wall sliding into the cut along the northward-dipping bedding planes.

As a result of this failure extensive design modifications were made (Figure A.51) Highway grades were increased to permit a reduction in the depth of the excavation.

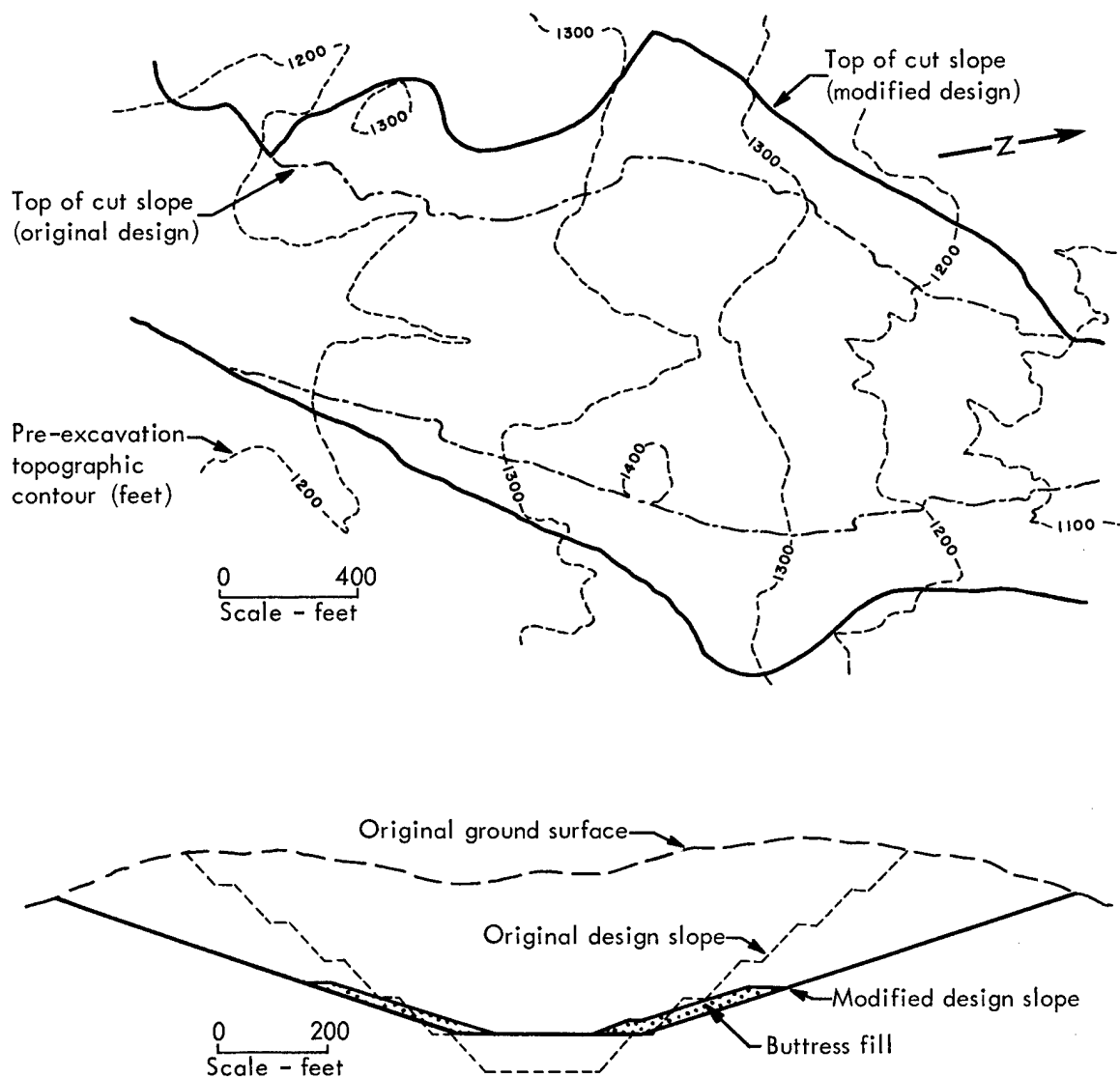


Figure A.51 Mullholland Cut; plan and representative profile.
(Adapted from Reference 232, Smith and Cedergren, 1962)

Benched 35° slopes were abandoned in favor of unbenched 19° slopes. To provide still further assurance of stability, the lower one-third of these flattened slopes was buttressed with a 50- to 75-foot thickness of fill, which was also sloped at 19°.

Reference 121

NEW CORNELIA MINE AJO, ARIZONA (51)

This mine is located in a desert plains region at the eastern extremity of the Little Ajo Mountain in southwestern Arizona. The local topography is mature and somewhat hilly, with elevations ranging from about 1,700 to 3,000 feet. The hills of the area rise abruptly above broad pediments which grade into an alluvial intermontaine plain. The region is arid, with average annual precipitation totalling about 9 inches. Temperatures in the area range from a winter minimum of 17°F to a summer maximum of 115°F.

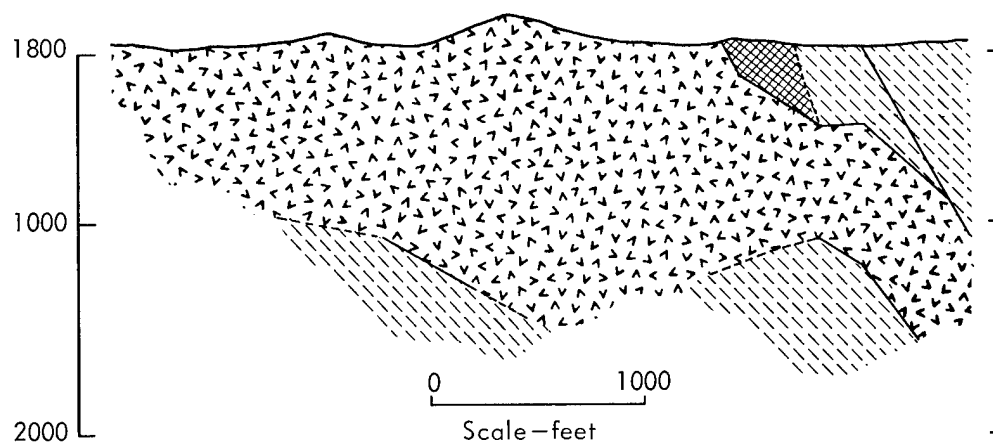
The New Cornelia Pit has been excavated to a depth of 700 feet in a porphyritic quartz monzonite stock which has intruded and domed earlier rhyolitic lavas and pyroclastics (Figure A. 52). Mineralized portions of the intrusive are intensely shattered and altered (silicified or sericitized). The silicified monzonite is hard and structurally competent (probably more so than the unaltered rock).

Several major faults trend north-south and NE-SW across the pit area. The Charlie fault, striking across the central portion of the pit in a generally N-S direction, and dipping eastward at 40 to 65°, is the locus of particularly intense brecciation (Figure A.53).

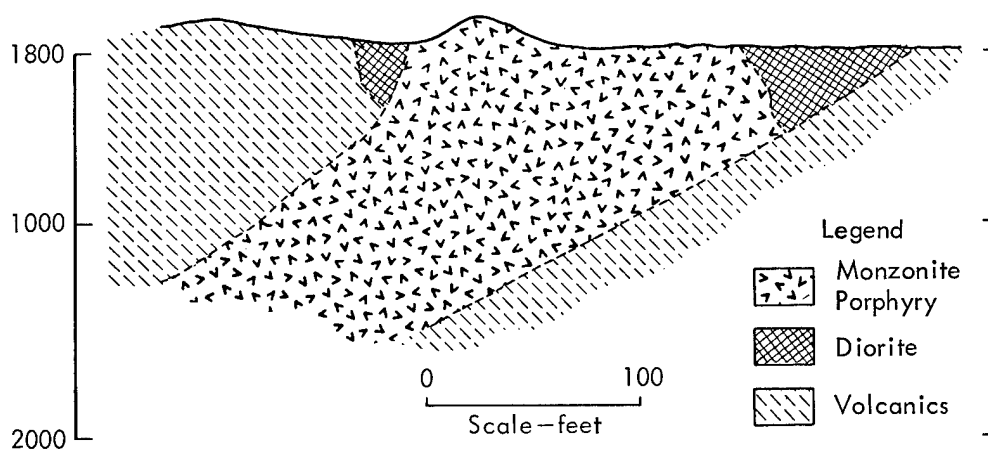
Pit slopes are benched at 40-foot vertical intervals throughout most of the pit. This height is dictated by the size of the electric shovels in use. The south wall of the pit is benched at 33-foot vertical intervals because of the type of drilling equipment used in that area. Operating benches are generally sloped at 56°, and final bench slopes are steepened to 70°. Overall operating slope angles range from 14 to 22°.

An ultimate pit depth of about 1,000 feet is planned. Ultimate overall pit wall slopes will be 45° in competent materials and 37° in less competent materials.

References 8, 34, 41, 42, 48, 58, 60, 109, 136, 187 (cf. Figure 3.9)



Geological section through ore body, looking northeastward



Geological section through ore body, looking northwestward

Figure A.52 New Cornelia Mine; representative geological sections.
(Adapted from Reference 58, Ingham and Barr, 1932)

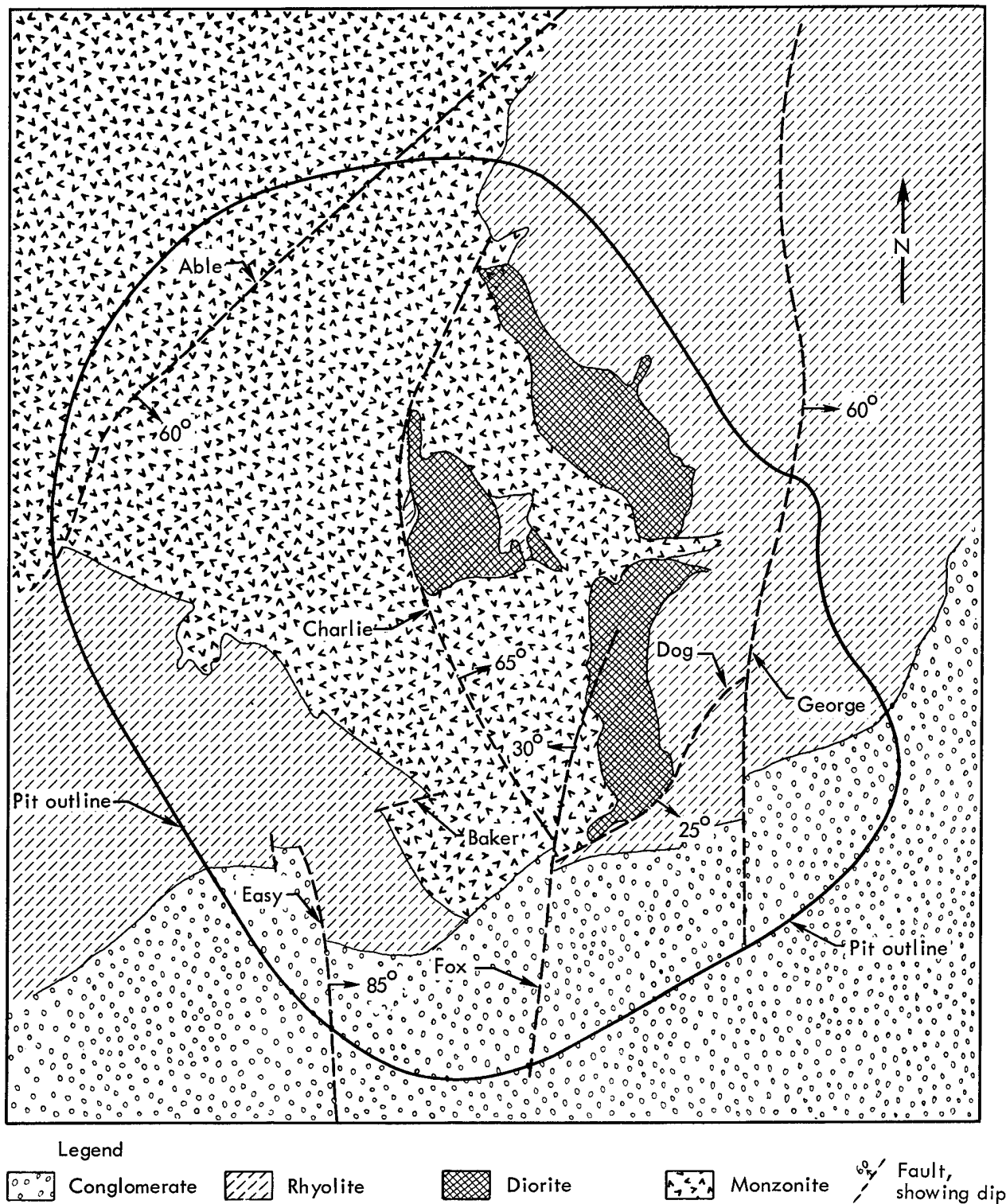


Figure A.53 New Cornelia Mine; generalized geological map. (Adapted by permission from Reference 134, "Geology of the Porphyry Copper Deposits -- Southwestern North America," edited by Titley and Hicks, The University of Arizona Press, Tucson, copyright 1966)

PEARL HANDLE AND WEST PITS RAY, ARIZONA (55, 74)

These open pit copper mines are located near the town of Ray in east-central Arizona (Figure A.54), approximately 75 miles east-southeast of Phoenix and 70 miles north of Tucson. The deposit lies within the Mineral Creek mining district, near the western margin of the Mexican Highland section of the Basin and Range physiographic province. It occupies the slopes of the Tortilla and Dripping Springs Mountains and the floor of the intervening valley. The mountain ranges of the area are paralleled by major fault systems, and the valleys represent structural depressions which have been filled with detrital material from the adjacent uplands. Elevations within the district range from less than 2,000 feet to more than 5,000 feet. The climate of the region is arid.

Bedrock formations of the Ray area include igneous, metamorphic, and sedimentary rocks which have been covered by younger limestones and quartzites. This entire sequence has been transected by a series of intrusives ranging in composition from acidic to intermediate. These, in turn, are overlain by conglomerate and pyroclastics (Figures A.55 and A.56).

Major fault systems are the dominant structural features of the Ray area. Most of these faults are normal, although reverse and thrust faults are not uncommon. The dominant fault system parallels the NNW-SSE trend of the major topographic features, while a subordinate system crosses the area nearly at right angles to this primary trend. Most of the igneous intrusives of the area have been localized along, and are bounded by, major fault systems.

The first recorded mining activity in the region began in 1870. Interest was first shown in the low-grade copper deposit during the early 1900's. Conversion to open pit operations started in 1948 and was completed by 1955. At present, ore is being mined from two open pits (Pearl Handle and West pits).

Ore grade mineralization is chiefly localized in sedimentary and metamorphic rocks adjacent to a porphyritic quartz monzonite intrusive. The mineralized zone is bounded on the north, east, and west by faults, and on the south by a contact between igneous and metamorphic rocks. The ore deposit which is being mined in the Pearl Handle and West pits occurs as an irregular blanket several hundred feet thick.

The West Pit has been excavated in mineralized portions of the Pinal schist. The pit is roughly triangular in plan, with sides ranging from about 1,900 to 2,200 feet in length. Overall pit wall angles range up to 45°. In 1959, a large slide occurred on the west side of the pit; a vertical displacement of about 75 feet occurred at the top of the slide, and bulging at the toe reached a maximum height of 10 feet. At present, cracks can be found as far as 150 feet back from the pit rim in this same area. Excavation in this portion of the pit has continued since the slide occurred, but flatter slopes have been employed.

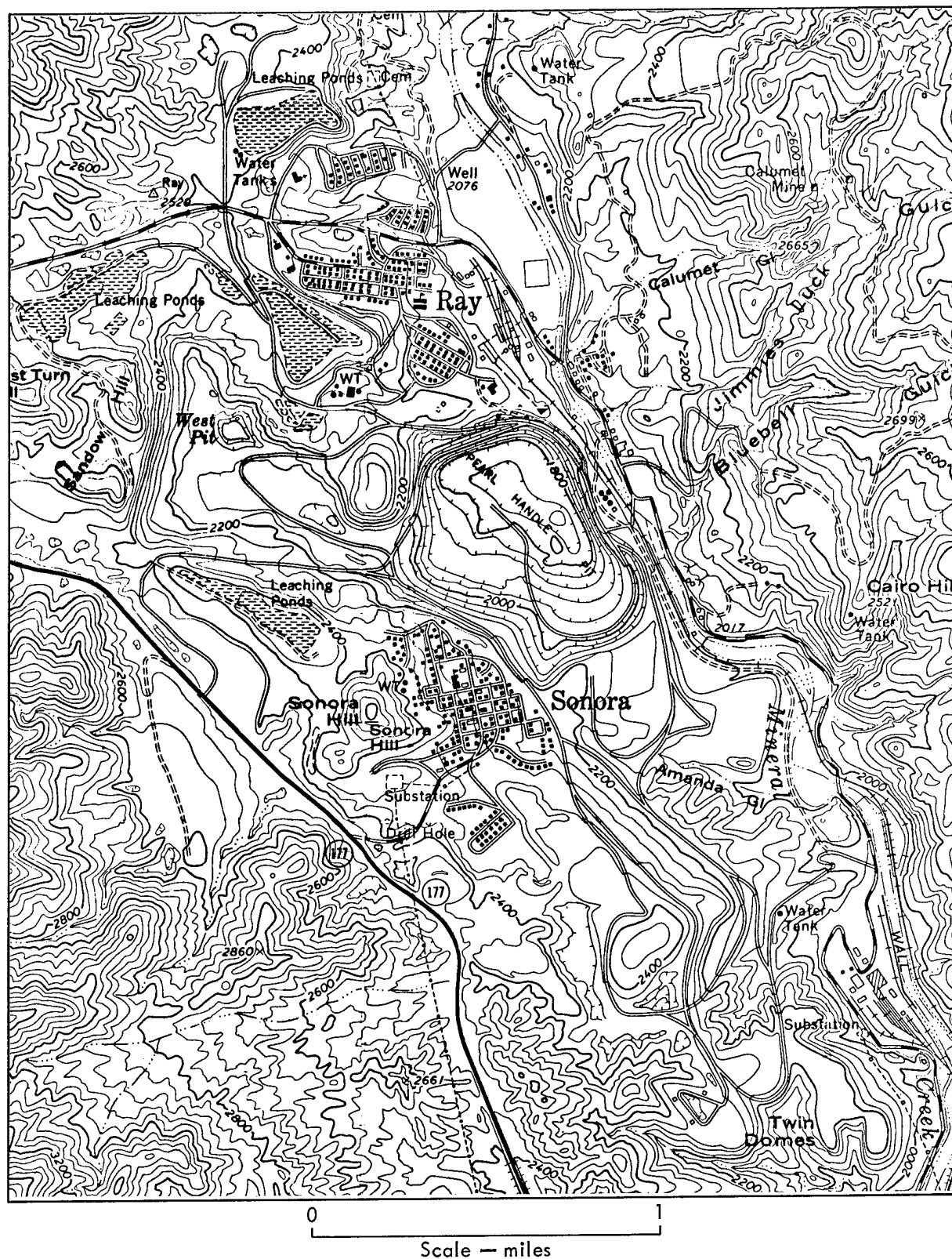


Figure A.54 Topographic Map; vicinity of Pearl Handle and West Pits.
(Adapted from U. S. G. S. Topographic Map, Ray Quadrangle)

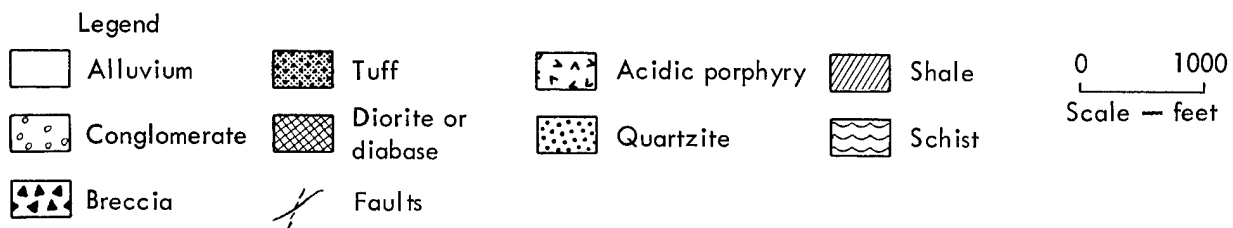
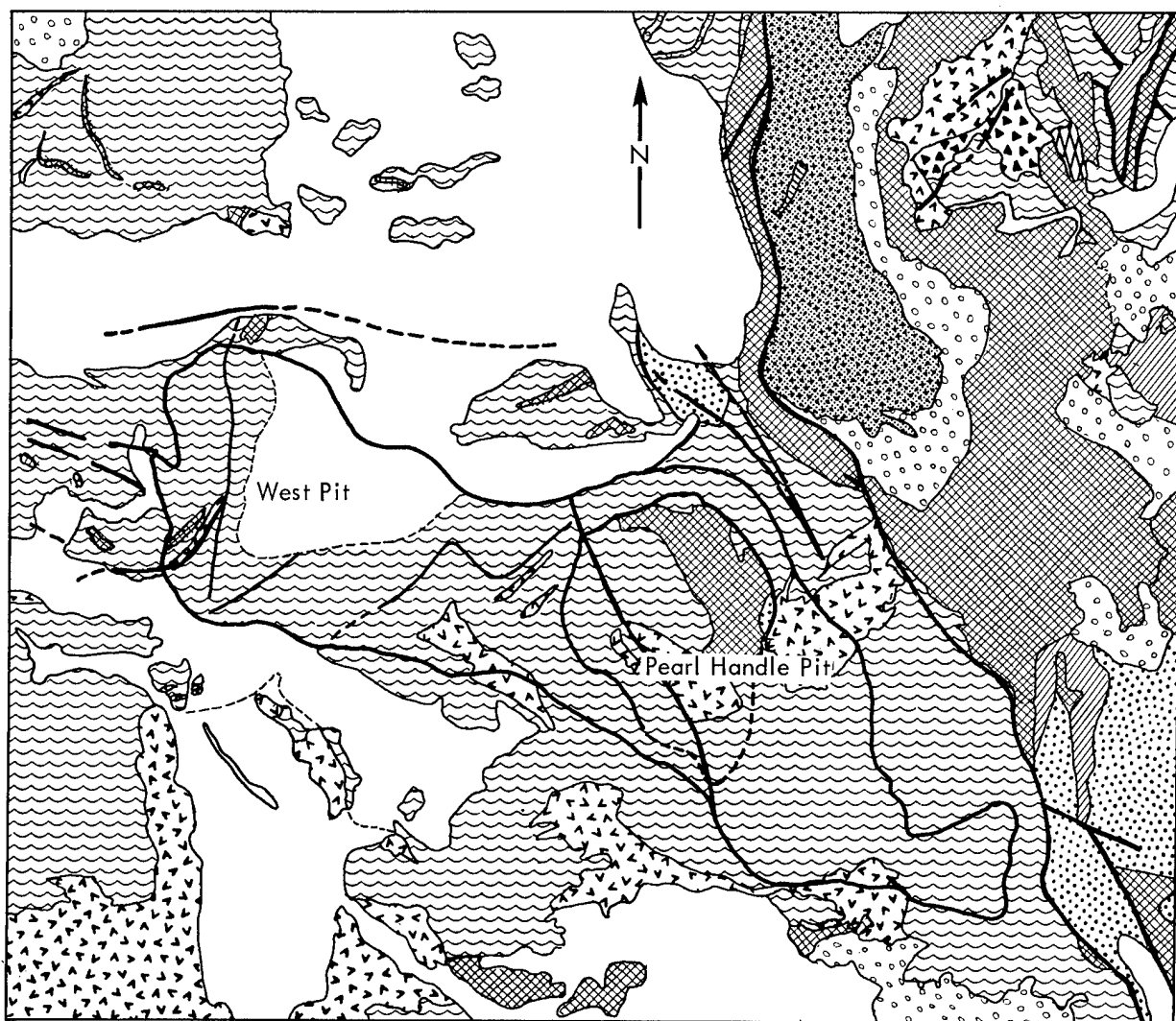


Figure A.55 Generalized geological map; vicinity of Pearl Handle and West Pits.
(Adapted by permission from Reference 134 "Geology of the Porphyry Copper Deposits — Southwestern North America," edited by Titley and Hicks, The University of Arizona Press, Tucson, copyright 1966.)

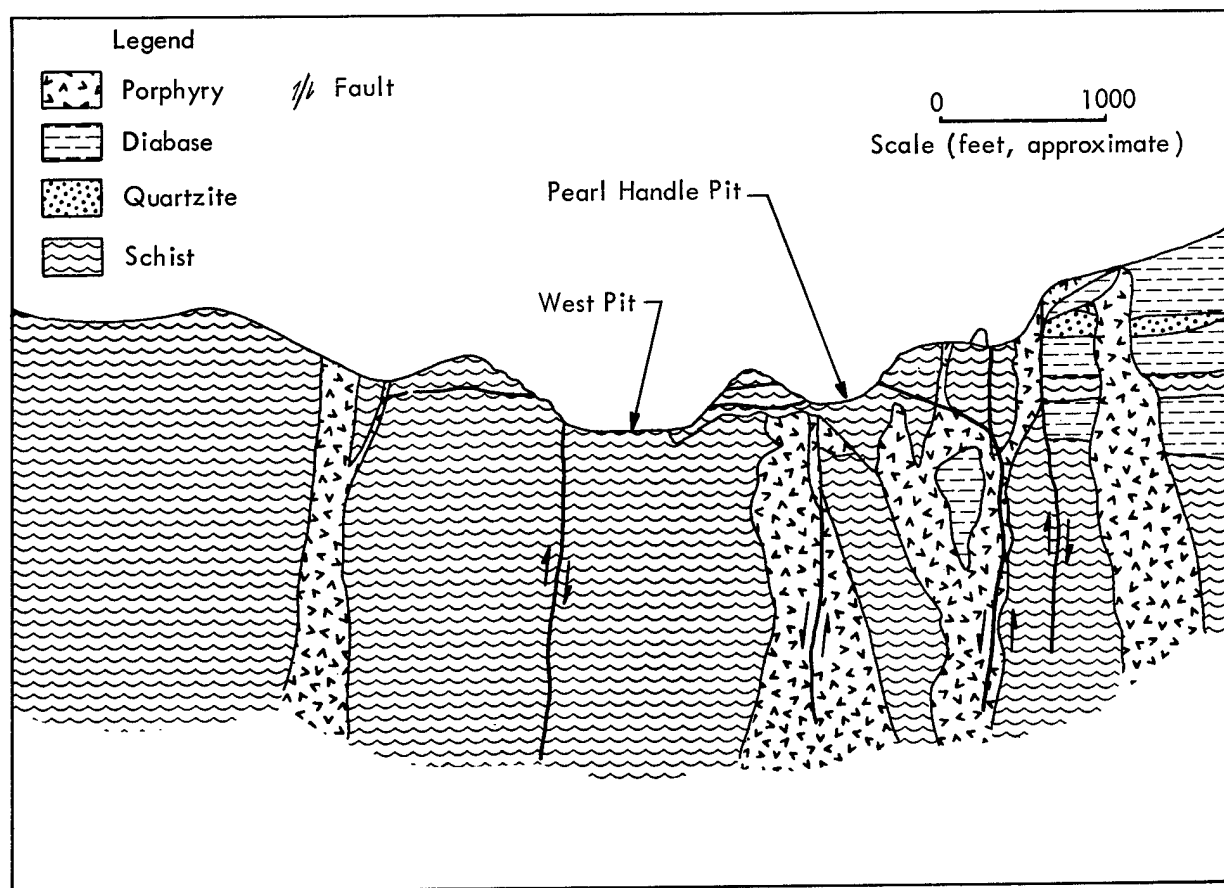


Figure A.56 Generalized geological section through Pearl Handle and West Pits. (Adapted by permission from Reference 134, "Geology of the Porphyry Copper Deposits -- Southwestern North America," edited by Titley and Hicks, The University of Arizona Press, Tucson, copyright 1966)

The Pearl Handle Pit has been largely excavated in mineralized portions of the Pinal schist and adjacent porphyritic intrusives. Small areas of shale and quartzite have also been encountered in the excavation. The pit is roughly rectangular in plan, with a length of about 3,400 feet and a width of about 2,400 feet. Maximum relief of about 570 feet is on the northwest side. Average slope angles range from 17 to 45°. Continuous slopes range up to about 950 feet in length (Figure A.57). In late 1954 or early 1955, a major slide occurred on the southwestern side of the pit. The slide movement was apparently localized along the Emperor fault which dips toward the excavation at 37°. The preslide slope was approximately 300 feet long with an average slope of about 45°. Vertical displacement at the top of the slide was about 50 to 60 feet.

References 8, 24, 67, 76, 90, 91, 92, 100, 131, 179

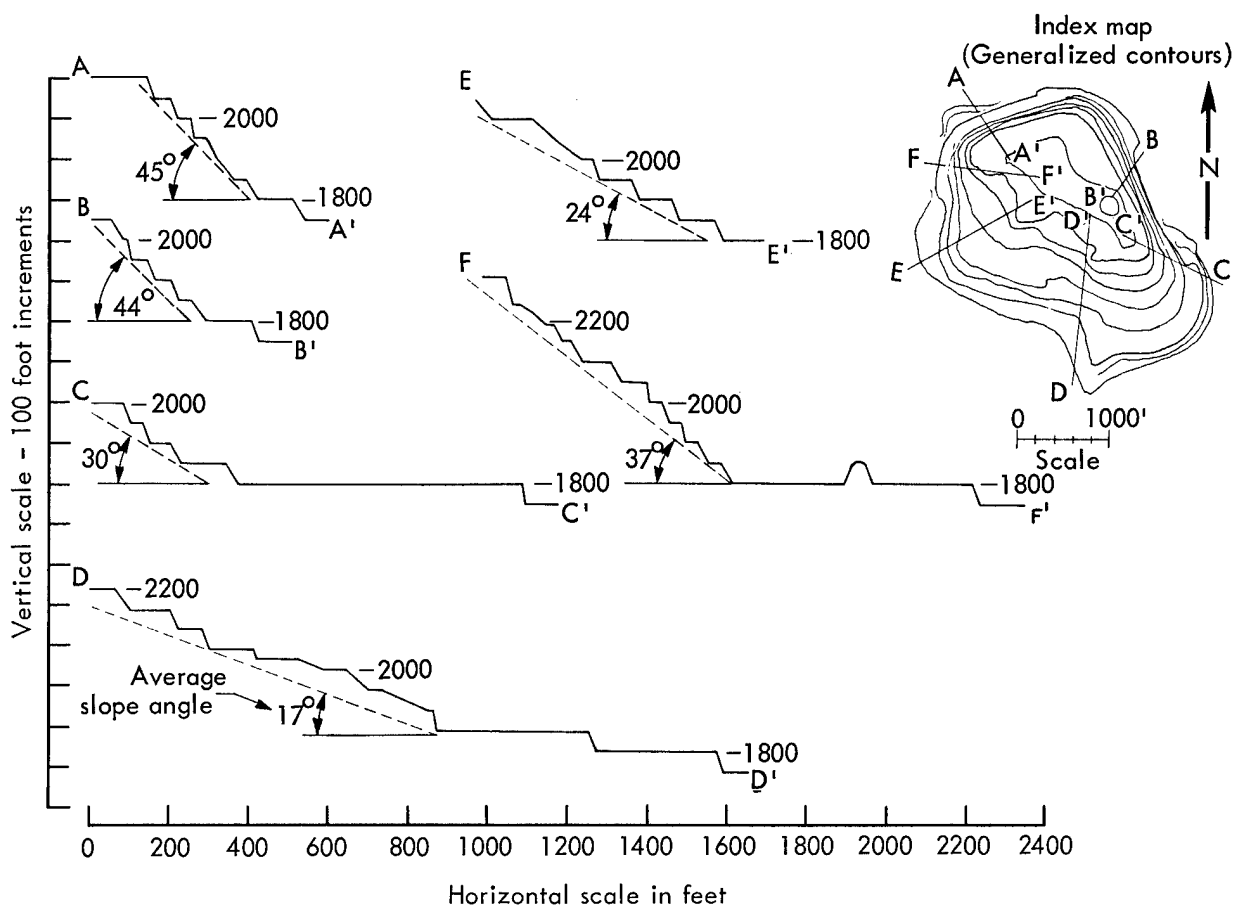


Figure A.57 Pearl Handle Pit; generalized plan and representative profiles.
(Adapted from data contributed by Kennecott Copper Corporation)

SANFORD LAKE MINE TAHAWUS, NEW YORK (61)

The Sanford Lake Mine at Tahawus, New York, has been excavated in a conformable sequence of gabbro, anorthosite, and massive magnetite-ilmenite lenses which strike NE-SW and dip northwest at angles of approximately 45° (Figures A.58 and A.59). A 50-foot-wide fault zone, which is exposed along the northwestern wall of the pit (Figure A.60) also strikes NE-SW, but dips northwest at about 75°.

The pit is oval in shape, being elongated parallel to the strike of the magnetite-ilmenite ore bodies. It measures approximately 2,700 feet in length and 1,000 feet in width, and has been excavated to a maximum depth of 700 feet.

Working faces within the pit are now broken at 44-foot vertical intervals with established permanent benches 40 feet in width at 132-foot intervals. The faces of

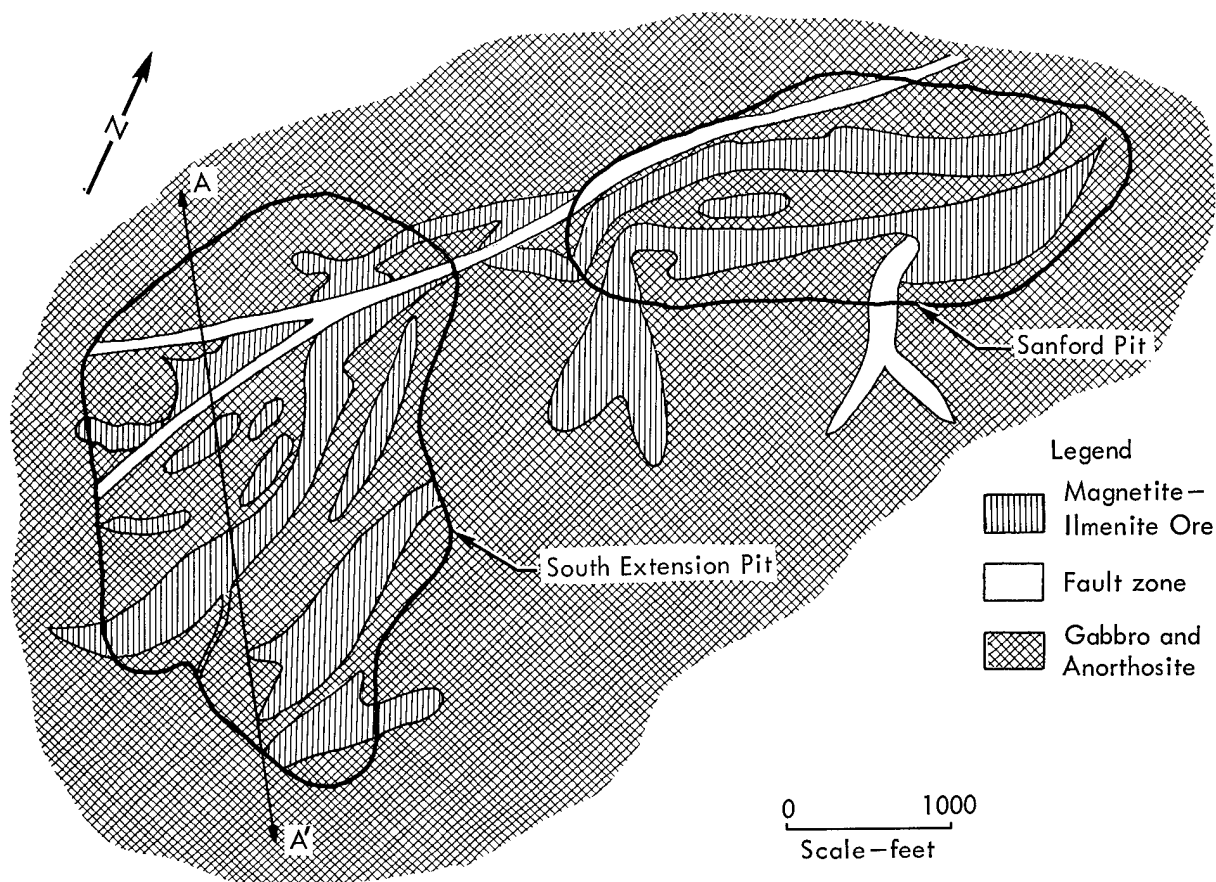


Figure A.58 Sanford and South Extension Pits; generalized geological map.
(Adapted from data contributed by National Lead Company)

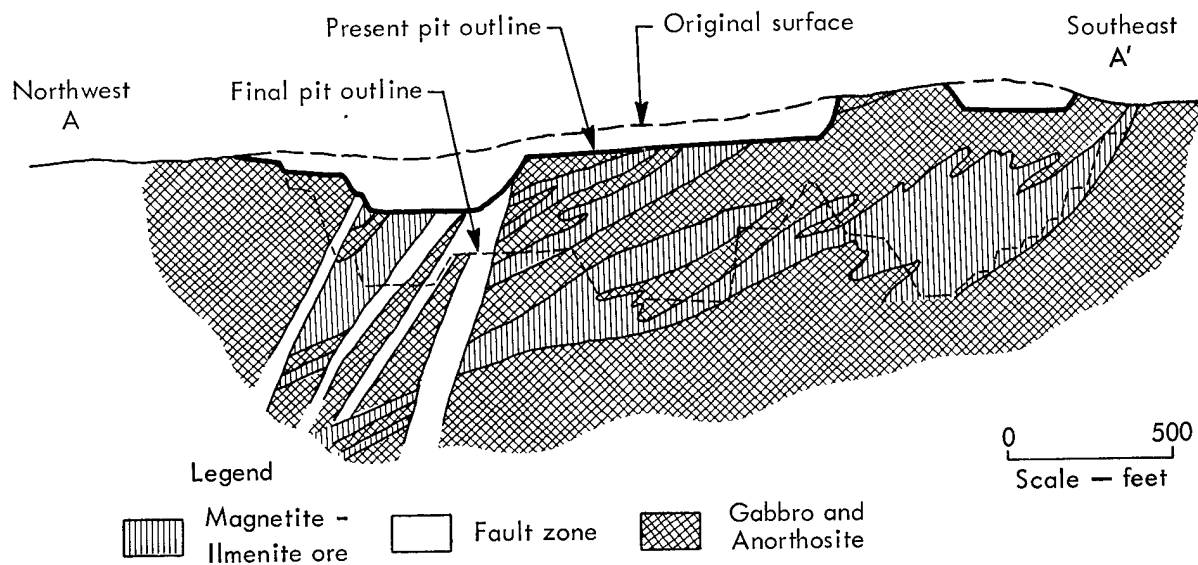


Figure A.59 South Extension Pit; profile and geological section.
(Adapted from data contributed by National Lead Company)

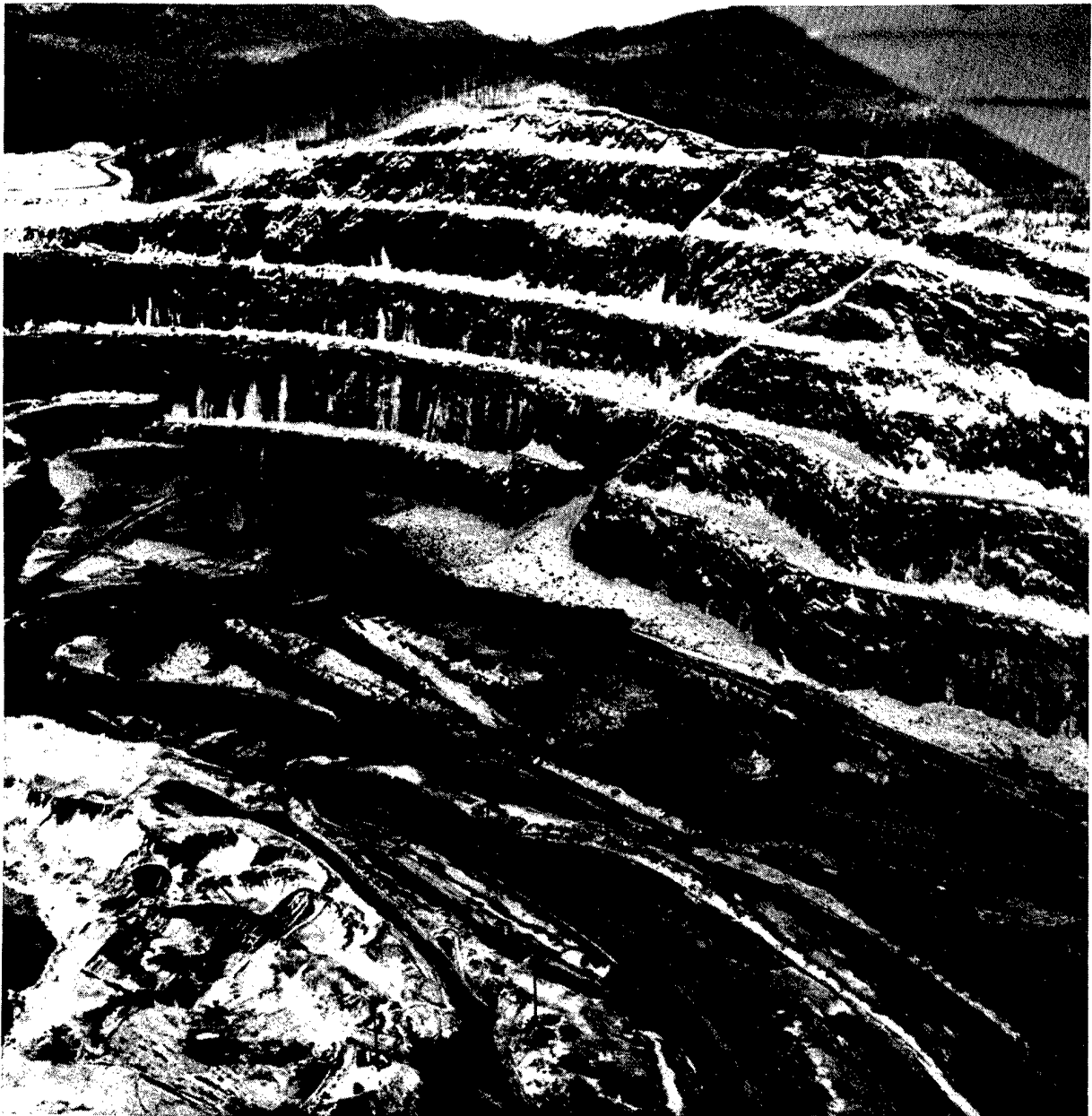


Figure A.60 Sanford Pit; showing branching fault in southeastern wall.
(Contributed by National Lead Company)

operating benches are sloped at 80° and permanent benches at 73° . The overall declivity of the pit wall is about 50° (Figure A.61).

Some slope stability problems were encountered in the vicinity of a fault zone in the southern wall of the pit, despite the fact that the pit wall, at this locality, is approximately perpendicular to the strike of the fault zone. On the other hand, no significant slope stability problems were encountered along the northwestern portion of the pit wall, despite the presence in that area of a broad fault zone which parallels the pit wall and dips steeply away from the pit.

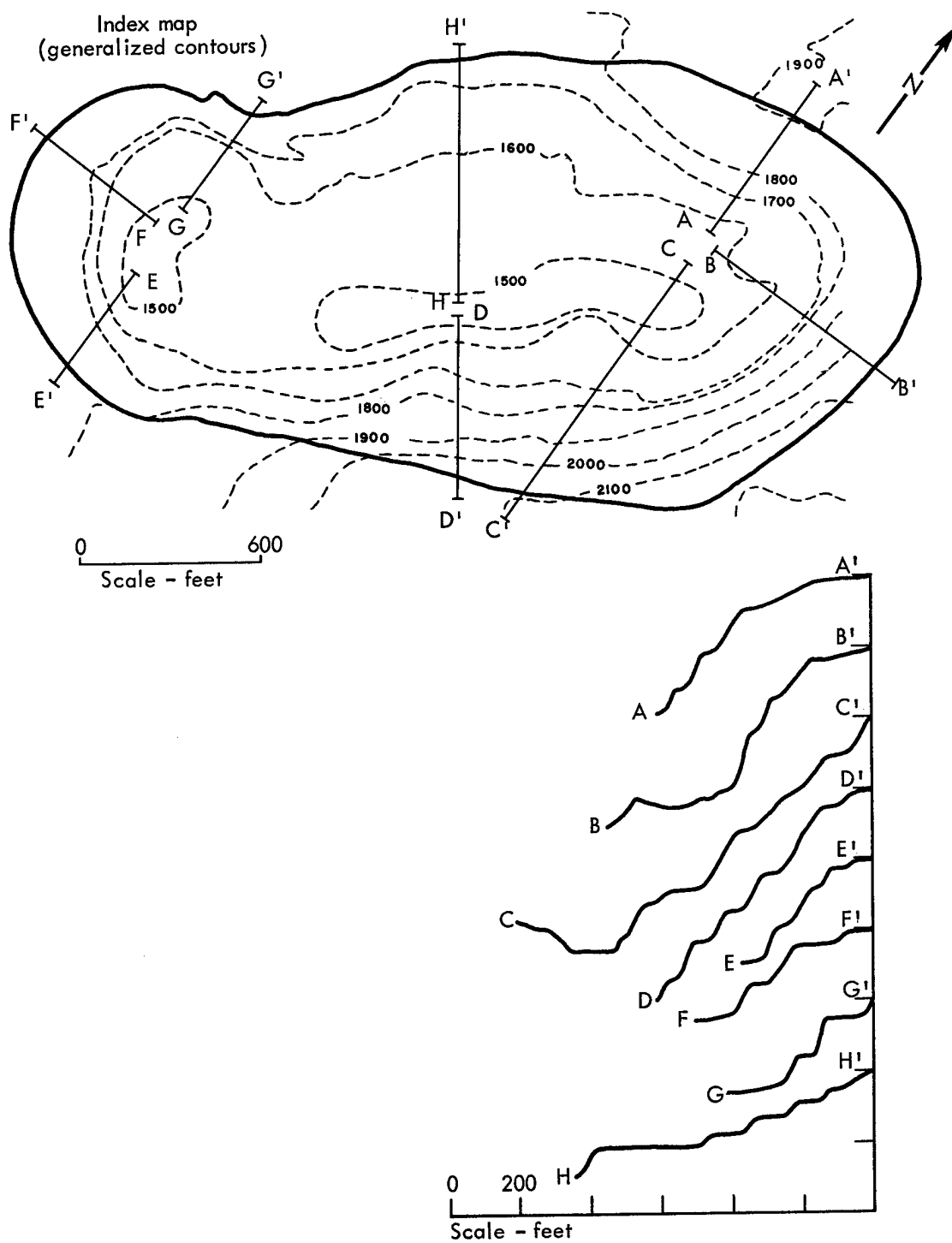


Figure A.61 Sanford Pit; generalized plan and representative profiles.
(Adapted from data contributed by National Lead Company)

The pit is located in a region of humid temperate climate having a mature topography and moderate (about 500 foot) local relief. The original level of the water table was essentially coincidental with the bedrock surface, but this has been lowered substantially by the pumping of water from the pit.

A second pit is currently under development to the southwest of the original Sanford Lake Pit. This mine is being excavated in a very similar structural, lithologic, and topographic setting; plans for pit wall development are being based upon the configurations successfully employed in the older pit. The new pit will have a maximum length of about 2,800 feet and a maximum depth of about 450 feet.

References 2, 182

STEEP ROCK LAKE DISTRICT ATIKOKAN, ONTARIO (66)

The iron ore of this district is localized within a steeply (62 to 77°) pitching body consisting of a soft incompetent mass of fine-grained quartz, pyrolusite, and kaolinite, with included subangular fragments of chert, hematite, and goethite.

Initial excavations in this locality involved the removal, by dredging and hydraulicking, of up to 150 feet of varved lacustrine silts and clays. Slopes in these fine-grained sediments were broken at 20-foot vertical intervals by benches which increased in width (from a minimum of 50 feet) with depth. The benches were sloped at 18°, approximating the steepest natural slopes encountered in the unwatered lake bed. Although a few flow slides occurred in these slopes, they were interpreted as resulting from some artificially induced disturbance of the sediments rather than as a reflection of inherent angular instability.

A thick accumulation of gravel and boulder clay underlying the lacustrine sediments was excavated to an average slope of 34°, using intermediate benches sloped at 40 to 50°

A limestone formation which forms the footwall of the ore body and the mafic pyroclastic "ash rock" of the hanging wall have been excavated in irregularly benched slopes up 550 feet in height and having an average declivity of 55° (Figure A.62).

References 7, 8, 59, 168.



Figure A.63 Toquepala Mine; looking northward.
(Contributed by Southern Peru Copper Corporation)

Walls of the pit are broken at 49-foot vertical intervals by benches ranging in width from 76 to 113 feet. Individual banks are sloped at 70° , and overall pit slopes at the present time are about 25° (Figure A.65).

References 54, 105, 149, 192

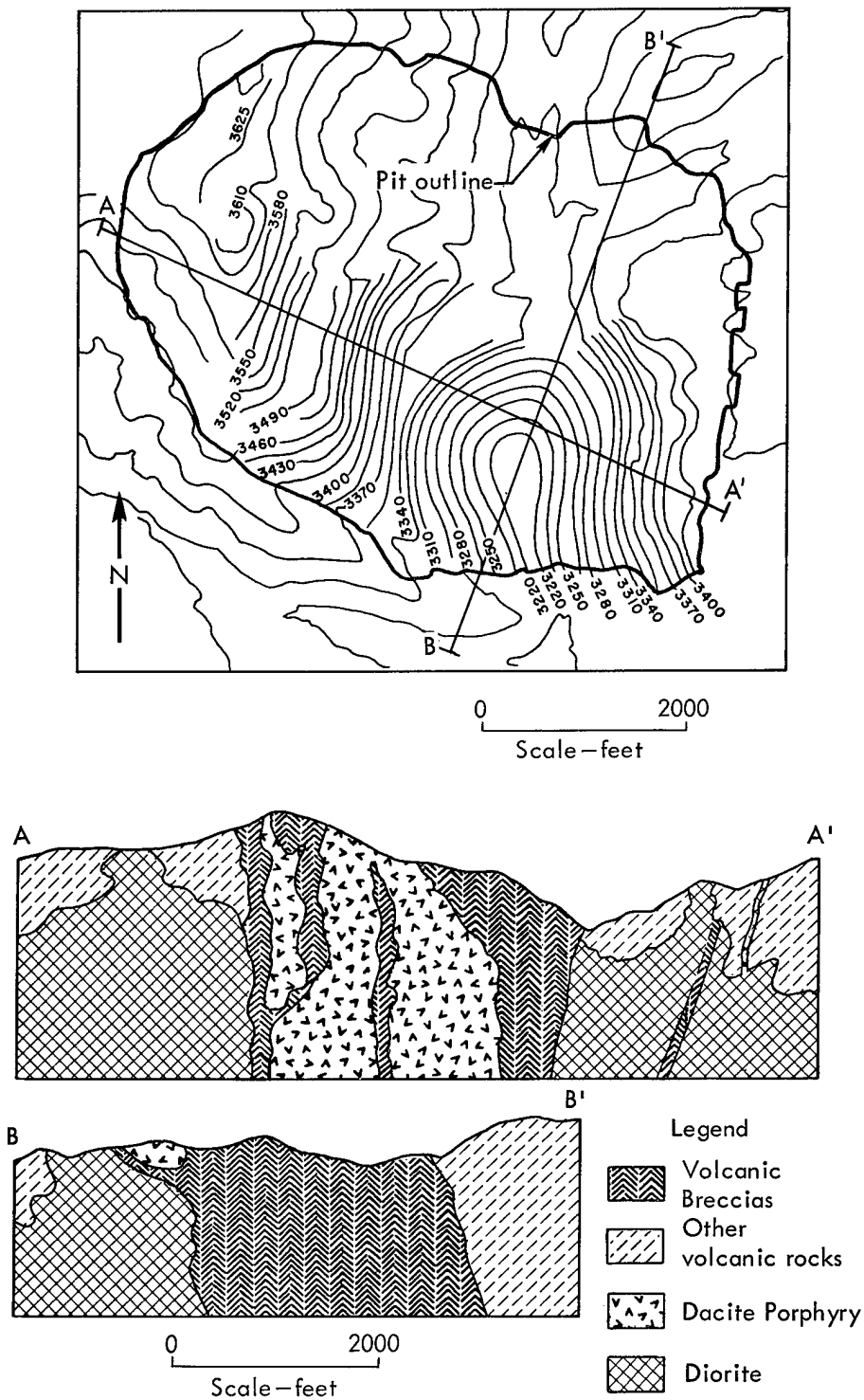


Figure A.64 Toquepala Mine; generalized plan and representative geological sections. (Adapted from Reference 105, Richard and Courtwright, 1958 and from data contributed by Southern Peru Copper Corporation)

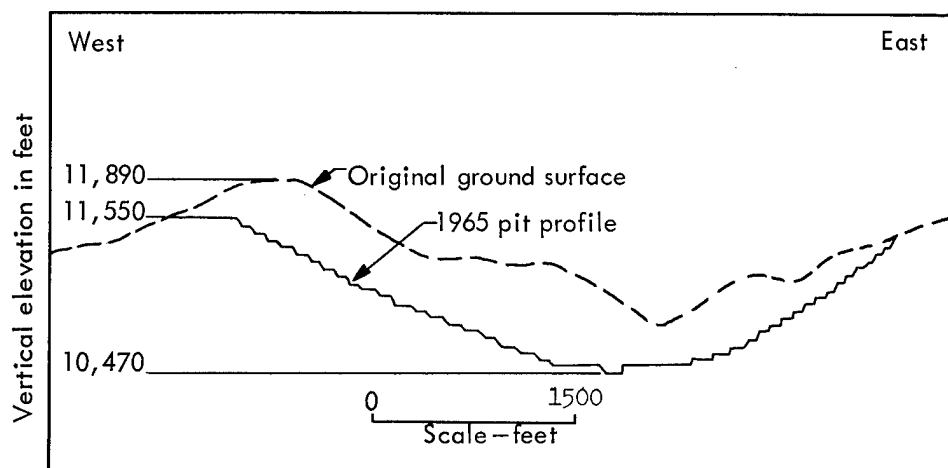


Figure A.65 Toquepala Mine; representative profile showing progress of excavation.
(Adapted from data contributed by Southern Peru Copper Corporation)

UNITED VERDE PIT JEROME, ARIZONE (71)

This open pit copper mine is located at an elevation of 5,490 feet on the eastern slope of a 7,000-foot mountain range in central Arizona. The pit has been excavated to a depth of 380 feet in a massive sulfide "pipe" which has replaced a chloritic schist selvage separating dioritic and acidic porphyry intrusives that form the hanging wall and footwall, respectively, of the ore deposit (Figure A.66). The mineralized pipe pitches westward at an angle of approximately 45°.

Initial pit development plans called for a 45° angle on the footwall slope of the pit (following the footwall contact of the ore body) and a 63° angle in the diorite and ore which form the hanging wall slope of the pit.

In 1929, after two years of open pit mining activity, subsidence within the ore body (resulting from the collapse of old underground stopes) resulted in fracturing of the hanging wall diorite. A subsequent (1941) slide proved that the 63° hanging wall slope was unstable, and it was decided to reduce this slope to 45° above the 160-foot level (Figure A.67).

Mining operations at United Verde were terminated in 1953.

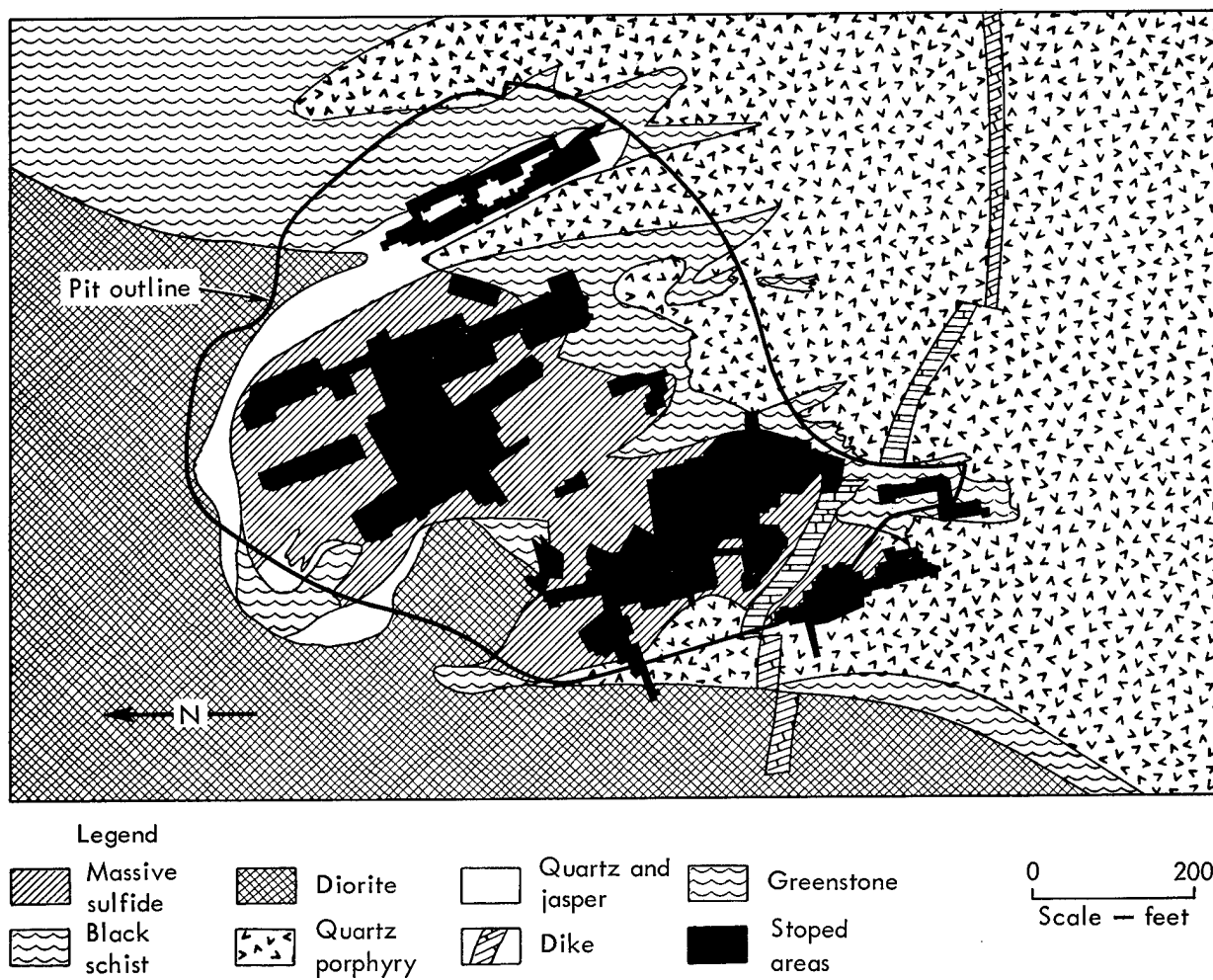


Figure A.66 United Verde Pit; generalized geological map.
(Adapted from Reference 1, Alenius, 1930)

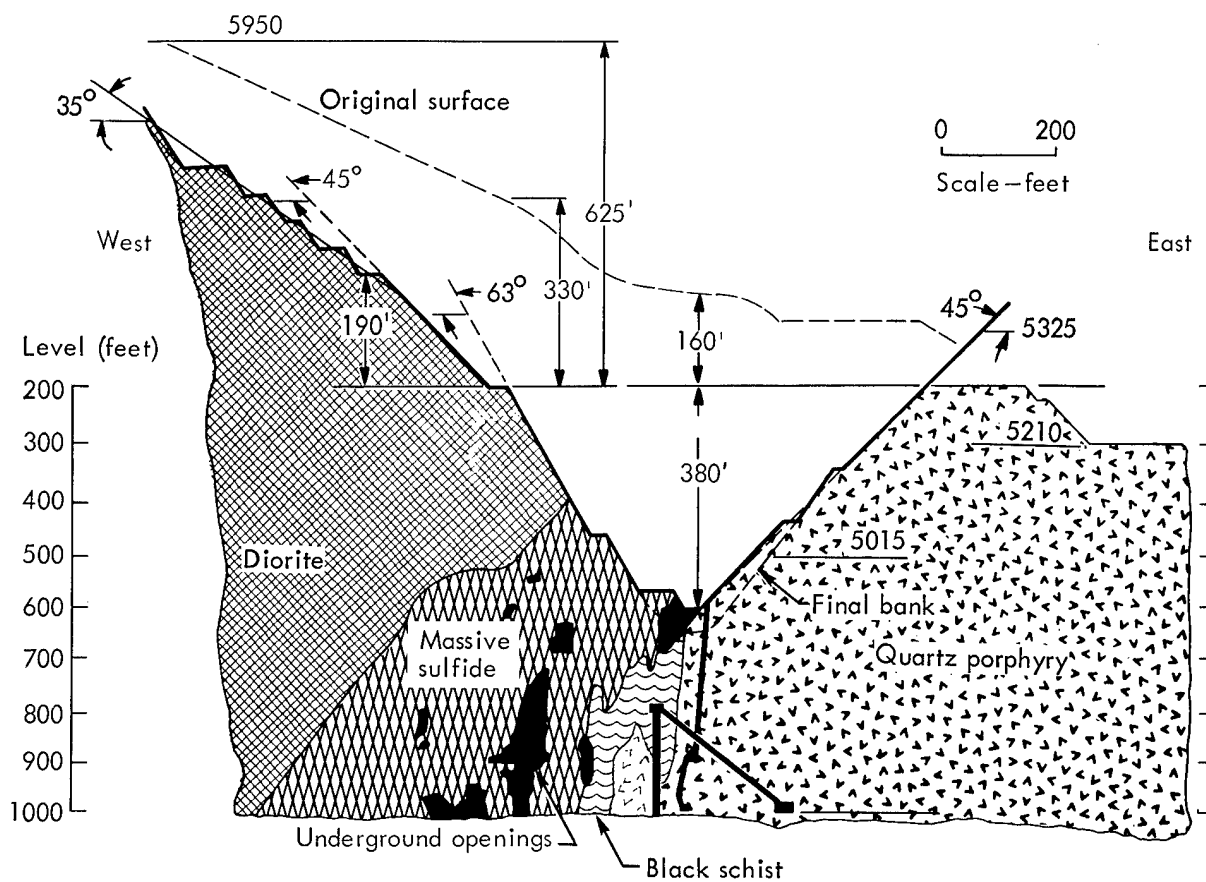


Figure A.67 United Verde Pit; representative profile and generalized geological section. (Adapted from Reference 85, Peele, 1948 by special permission of John Wiley & Sons, Inc.)

APPENDIX B
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I. DESCRIPTIONS OF SPECIFIC LOCALITIES

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164. Atlas Corporation (62)
165. Bagdad Copper Corporation (2)
166. Basic Incorporated (3)
167. Bethlehem Copper Corporation, Ltd. (19,27)
168. Caland Ore Company, Ltd. (66)
169. Canadian Johns-Manville Co., Ltd. (26)
170. Compania Minera de Cananea S. A. de C. V. (64)
171. Duval Corporation (22)
172. Empresa Minera de Mantos Blancos S. A. (57)
173. Esso Standard Oil Corporation (78)
174. H. E. Fletcher Company (82)
175. Hanna Mining Company (8,44,65)
176. Inspiration Consolidated Copper Company (36,67)
177. Johns-Manville Products Corporation (86)

178. Kaiser Steel Corporation (18)
179. Kennecott Copper Corporation (5, 30, 35, 55, 70, 72, 74)
180. Luossavaara-Kiirunavaara Aktiebolag (31)
181. Miami Copper Company (9)
182. National Lead Company (61)
183. Nchanga Consolidated Copper Mines, Ltd. (50)
184. Nevada Porphyry Gold Mines, Inc. (58)
185. Oesterreichisch-Alpine Montangesellschaft (21)
186. Ottawa Silica Company (89)
187. Phelps Dodge Corporation (51)
188. Pittsburgh Pacific Company (40)
189. Roan Selection Trust, Ltd. (10)
190. Rock of Ages Corporation (92)
191. J. R. Simplot Company (39)
192. Southern Peru Copper Corporation (69)
193. United States Borax and Chemical Corporation (6)
194. Wells-Lamson Quarry Company, Inc. (94)
195. Western Nuclear, Inc. (17, 23)
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